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TRENDS IN PRACTICE AND COSTS OF CONCENTRATING COPPER ORES



BY

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By T. G. Chapman^{2/}

INTRODUCTION

This paper has been abstracted from a forthcoming Bureau of Mines bulletin on the concentration of copper ores in North America.

GENERAL DEVELOPMENTS IN CONCENTRATING
COPPER ORES FROM 1912 TO 1932

During the past 20 years the milling of copper ores as practiced in the larger concentrators has changed to such an extent that comparatively few of the machines in use at the beginning of this period remain in service today.

Concentrating Practice in 1912

In 1912 the usual equipment for concentrating copper ores comprised jigs for the coarser sizes, tables and sand vanners for the intermediate sizes, and vanners for the finer crushed and ground material. Rolls operated in series under graded crushing conditions, were in common use for the liberation of the copper minerals; grinding by Chilean or Huntington mills ordinarily was confined to the coarser tailings of jigs and tables.

The recovery of copper contained in the feed of most concentrators was low, ranging from 60 to 75 percent. The losses of copper in the tailings comprised unliberated copper minerals attached to or enclosed within gangue particles and finely divided copper minerals free from gangue but too finely divided to be saved by the concentrating machines then in use.

1/ The Bureau of Mines will welcome reprinting of this paper provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6792."

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Improved Methods of Treating Slimes by Gravity Concentration

The period just before the introduction of flotation - 1911 to 1914 - was one of intensive study and test-plant work to reduce the losses of copper in tailings. At Anaconda, after thorough study of the slime problem, the round table was selected as the machine best adapted to decrease the copper losses in the slime product. A round-table plant was erected and put into operation in March 1914.^{3/} The slimes going to the round tables contained about 2 percent of copper of which about 50 percent was recovered. The copper content of Anaconda tailings, other than slime tailings, at this period was about 0.60 percent. A leaching process following roasting was devised to treat this material.

At Miami considerable experimental work was done during 1911 and 1912 to reduce losses in tailings by grinding the coarser sand-table tailing in pebble mills and treating fine slime material on multiple-deck tilting slime tables.^{4/}

These examples have been given to indicate the attempts made during this period to overcome the defects of the gravity-concentration process, and although the recovery of both free and attached copper was improved by the methods described the gravity process was still inadequate to effect high recoveries of copper.

Introduction of Flotation Process

The flotation process was introduced in the larger copper concentrators from 1913 to 1916. Prior to this period the Elmore vacuum flotation process had been tried, and Macquisten tube flotation machines had been operated at Golconda, Nev., in 1906 and at the Morning mill at Mullan, Idaho, in 1911.^{5/} The first large-scale application of a process similar to present flotation methods in the United States was that of J. M. Hyde^{6/} in 1912 at the 500-ton lead-zinc mill of the Butte & Superior Copper Co. During the 3 years which followed Hyde's successful commercial demonstration of the process, flotation methods were generally adopted in concentrators which treated sulphide copper ores.

Early Types of Machines Used

Hyde used a mechanical-agitation-type machine, and the first machines introduced in the copper concentrators were also of this type. The pneumatic-type machine was first developed by J. M. Callow and introduced in the concentrator of the National Copper Co. at Mullan, Idaho, in April 1914.^{7/} Although

^{3/} Laist, F., and Wiggin, A. E., The Slime-Concentrating Plant at Anaconda:

Trans. Am. Inst. Min. Eng., vol. 49, 1914, p. 470.

^{4/} Hunt, H. D., Milling Methods and Costs at the Concentrator of the Miami Copper Co., Miami, Ariz.: Inf. Circ. 6573 Bureau of Mines, 1932, p. 2.

^{5/} Hoover, T. J., Concentrating Ores by Flotation: Mining Mag., London, 1914, 2d ed., p. 15.

^{6/} Hoover, T. J., work cited, p. 17.

^{7/} Callow, J. M., Notes on Flotation: Trans. Am. Inst. Min. Eng., vol. 54, 1916, p. 4.

straight mechanical-agitation type of cell is still in use at a few copper concentrators, generally it has been modified to the subaeration or mechanical-air type to reduce the consumption of power or to obtain improved metallurgical results. The original stationary-mat pneumatic cell is still employed at many concentrators, but it has also been modified to the rotating-mat or matless-type machine to eliminate mechanical difficulties brought about by developments in flotation practice.

Growth of Process as Applied to Copper Ores

Flotation was first introduced in copper concentrators as an accessory process to gravity methods; its usual role was that of treating the slime portion of the mill product, although at one or two plants it was first used to increase the grade of vanner concentrates. The immediate object of the introduction of flotation was therefore to reduce the losses of fine free mineral in slime products. The pebble mill had replaced Chilean grinding mills at a few concentrators before the introduction of flotation methods. The Door classifier had been introduced in gold-milling grinding circuits prior to the introduction of successful flotation operations.^{8/} The development of pebble-mill grinding in closed circuit with drag classifiers in concentrators treating copper ores produced an increasingly fine product for concentration with the result that the flotation process, although still accessory to gravity methods, obtained an increasingly larger proportion of the concentrator feed for treatment. The next step was the substitution of cast-iron or cast-steel balls for pebbles from 1914 to 1918. The development of ball-mill grinding again made possible an increase in the fineness of grinding, which in turn gradually resulted in flotation being the major process; gravity methods were confined to operation of tables as an adjunct to flotation treatment until the period 1923 to 1927, during which all-flotation methods were adopted at most concentrators. It should be noted, however, that all-flotation methods were adopted at some plants as early as 1914 to 1918, but the general elimination of tables occurred from 1923 to 1927; the comparatively few concentrators that retained tables after 1927 did so for reasons peculiar to each plant.

Selective Flotation

The elimination of tables made possible the next development in the progress of flotation, namely, the depressing of pyrite in the tailing with a consequent increase in the grade of concentrate produced. The fact that pyrite could be depressed successfully in the treatment of Cananea ore was known as early as 1921^{9/} but was not used until about 1924. The depressing of pyrite

^{8/} Dorr, J. V. N., The Dorr Hydrometallurgical Apparatus: Trans. Am. Inst. Min. Eng., vol. 49, 1914, p. 217; vol. 55, 1916, p. 678.

^{9/} Tye, A. T., Milling Methods and Costs at the Concentrator of the Cananea Consolidated Copper Co., Cananea, Sonora, Mexico: Inf. Circ. 6261, Bureau of Mines, 1930, p. 6.

was adopted at the Nacozari plant in 1923^{10/} and in Canada by the Britannia mill in 1923.^{11/} At first the depression of pyrite was effected in a lime circuit in which coal-tar and pine-oil reagents were used.

Reagents

A change in the kind of flotation reagents employed also helped lead to present practice. At the beginning of flotation operations the usual reagents comprised coal tar, creosotes, and pine oil, which were used in both acid and alkaline circuits. In general, the original collectors were either replaced by thio-carbanilide in orthotoluidin or retained until the almost general adoption of the lime, xanthate, and pine-oil circuit during the period 1924 to 1926. The benefits gained by the use of xanthate and similar reagents apparently depended somewhat upon the ore treated, but opinions differ in this respect. Some metallurgists hold that the adoption of the new reagents improved metallurgical results, whereas others believe that metallurgically the newer reagents did not improve results obtained with the older reagents but that the chief advantage derived from the new reagents was in the speed of floating the sulphide minerals which in turn resulted in greater capacities of flotation units with consequent reductions in cost. All-flotation practice and the depression of pyrite were introduced at some plants before the appearance of xanthate in copper-concentrator circuits, but the practice of eliminating tables and depressing pyrite became general after the introduction of the lime-xanthate-pine oil circuit.

Development of Matless and Rotating-mat Pneumatic Machines

The reasons for the introduction and development of the rotating-mat and matless types of pneumatic machines may be found in the mechanical difficulties formerly encountered at some concentrators which employed stationary-mat cells of the Callow type and depressed pyrite in lime circuits. The depression of pyrite caused the accumulation of heavy material on the mats; furthermore, the necessity of maintaining higher degrees of alkalinity with lime, for the depression of pyrite, increased the rate of deposition of lime salts in the pores of the mats. Both of these conditions resulted in reduced efficiencies of aeration, erratic metallurgical results, higher consumptions of power, and increased labor charges at many plants using the stationary-mat type machine and led to the adoption of the rotating-mat or matless types of pneumatic cells or of the mechanical-agitation machine.

^{10/} Rose, E. H., and Cramer, W. B., Milling Methods and Costs at the Nacozari Concentrator of the Phelps Dodge Corporation, Nacozari, Sonora, Mexico: Inf. Circ. 6358, Bureau of Mines, 1930, p. 6.

^{11/} Munro, A. C. and Pearse, H. A., Milling Methods and Costs at the Concentrator of the Britannia Mining and Smelting Co. Ltd., Britannia Beach, B. C.: Inf. Circ. 6619, Bureau of Mines, 1932, p. 4.

The rotating-mat cell was developed in the Nacozari concentrator, and the conditions which led to its development have been described by Rose and Cramer as follows:^{12/}

The MacIntosh flotation machine was originated and developed at Nacozari to meet conditions resulting from the changed physical character of tailings after the development of selective flotation. While the primary objective of the machine was to keep heavy sands and pyrite moving freely through the circuit, it was immediately perceived that the accomplishment of this purpose should automatically improve both mechanical and metallurgical efficiencies. The anticipated advantages did indeed materialize. Whereas the Callow machines, in which the entire mill feed was treated for nearly 3 years, required 3.75 kw.-hr. per ton of ore for blower air, the MacIntosh machines replacing them operated under identical conditions on the same feed and the same tonnage rate at an average power consumption of only 0.55 kw.-hr. per ton, including power both for air and for driving rotors. At the same time, metallurgy was very materially improved.

When the rotating-mat cell was first used at Nacozari two-ply unquilted canvas was used as the porous media, and the air pressure was 3 pounds per square inch. Rose and Cramer state further:

For the past 2 years, perforated gum rubber coverings have been used on the MacIntosh rotors. These are also a local development. They are high-grade gum rubber stock and have definite holes through them, 0.01 inch in actual, clear diameter, there being 180 such holes per square inch. These are rather more porous than canvas - in fact, when the sheets are held up to the light, one is able to see through them and distinguish objects on the other side. By means of a locally designed "porosity meter" the actual porosity of any medium is readily determined, and it has been found that these blankets pass as much air at 1 pound pressure as will canvas at 4 pounds pressure. Indeed, in practice they are able to operate at very low pressures, yet, as it has proved, without danger of back-filtration of solids. With these rubber blankets, the MacIntosh machines are operated at about $1 \frac{3}{8}$ pounds pressure per square inch, or just sufficient to give a comfortable margin in overcoming the hydrostatic head of the pulp in the machine.

^{12/} Rose, E. H., and Cramer, W. B., Milling Methods and Costs at the Nacozari Concentrator of the Phelps Dodge Corporation, Nacozari, Mexico: Inf. Circ. 6358, Bureau of Mines, 1950, p. 17.

The conditions at the Old Dominion concentrator which were responsible for the development of the Forrester matless cell have been described by Forrester and Cramer as follows:^{13/}

After the adoption of the all-flotation treatment of ores in 1924 with the use of lime and T-T mixture reagents it became evident that a change would have to be made in the machines using porous blankets. The blankets of these machines became clogged with lime salts in a short time, and extra men were required to pound the mats in order to keep them open. The rejected pyrite settled on the blankets, which further impeded aeration of the pulp. The air pressure was increased from 4 to 6 and 7 pounds per square inch, and even with this increase of pressure it was difficult to get sufficient air through the blankets for the proper aeration of the flotation pulps. It became necessary to change one blanket each day, and as there were 16 cells in operation the life of the blanket was 16 days. Furthermore, in order to obtain this 16-day life it was necessary to give each blanket an acid bath and a scrubbing every 4 days. Operating conditions at that time for pneumatic machines using porous blankets were therefore not satisfactory at the Old Dominion concentrator, and it was necessary either to adopt mechanical agitation machines or to devise some means of producing a froth other than by passing air through a porous medium. It was due to these difficulties that the Forrester method and machine were developed. The Forrester machine, as is generally known, is a matless-type pneumatic machine, the froth being produced by air at low pressure, introduced through 3/4-inch vertical parallel pipes spaced on 4-inch centers.

Although the tendency during the past few years has been toward replacing stationary-mat machines by newer pneumatic-type ones, the former have been retained at Hayden, Magna, Cananea, and Walkermine; at Anaconda and Macozari they have been retained for the treatment of slimes and at Calumet & Hecla and Engels for scavenging after initial treatment in mechanical-agitation machines.

Grinding and Re-treatment of Concentrates

In 1925 the Miami Copper Co. and in 1926 the Britannia Mining & Smelting Co., Ltd., adopted the practice of regrinding and re-treating, by selective-flotation methods, the concentrates obtained from a relatively coarsely ground product treated in bulk-flotation circuits. The reasons given for this practice are approximately the same for both concentrators. At each mill the ore treated is very low in grade, and the predominating copper mineral, although intimately associated with the pyrite, is fairly free from the gangue minerals after relatively coarse grinding. It was found at both plants that the depression of pyrite in the primary flotation operation resulted in poor

^{13/} Forrester, D. L., and Cramer, W. B., Milling Methods and Costs at the Concentrator of the Old Dominion Co., Globe, Ariz.: Inf. Circ. 6467, Bureau of Mines, 1931, p. 14.

recoveries unless relatively fine grinding of all the ore was practiced. Present methods used at these mills eliminate the expense of fine grinding of all the ore yet retain the advantages of high recovery of sulphide copper and high-grade concentrate.

In 1931 the New Cornelia mill of the Phelps Dodge Corporation started to regrind and re-treat concentrate, the regrinding operation liberating chalcoppyrite from gangue minerals to reduce the insoluble content of the concentrate.

Miscellaneous Developments

Although the changes made and already noted in the flotation treatment of copper ores comprising the adoption of all-flotation methods, the depression of pyrite, the use of the lime-xanthate-pine oil circuit, the development of the revolving-mat and matless types of pneumatic machines, and the grinding of bulk-flotation concentrates greatly affected metallurgical results and cost of concentrating, other developments in the milling process of less importance accompanied the major improvements. These include: (1) Improved classification which has resulted from the introduction of the bowl-type classifier; (2) simplification of crushing flow sheets by larger reduction ratios and production of finer crushing-plant products by the use of the cone crusher; (3) use of screens operated in closed circuit with the final stage of crushing which has produced a uniform maximum size of feed to the grinding mills with marked benefits in grinding operations; (4) substitution of balls for rods in the secondary grinding mills or in both primary and secondary grinding mills and increased ball- and rod-mill speeds which have resulted in increased capacities of grinding circuits; (5) regrinding of flotation middling before its return to the flotation cell which has become common practice and has resulted in improved grades of concentrate; (6) separation of primary slime followed either by the separate conditioning of the primary slime and ground product before their combined treatment in flotation machines or the separate flotation treatment of the primary slime and ground products which have resulted in improved conditioning of the flotation pulps and decreased losses of copper in tailing at some concentrators; (7) improved conditioning of ore pulps in the treatment of massive sulphide copper ores and reground bulk concentrate, especially the thorough aeration of these pulps prior to flotation, which has improved the speed of flotation operations when treating these products.

Developments in Treatment of Native Copper Ores

C. H. Benedict,^{14/} metallurgist of the Calumet & Hecla Consolidated Copper Co., recently described developments in the treatment of Lake Superior conglomerate ore as follows:

^{14/} Benedict, C. H., Milling Method and Cost at the Conglomerate Mill of the Calumet & Hecla Consolidated Copper Co.: Inf. Circ. 6364, Bureau of Mines, 1930, p. 3.

Lake Superior copper-concentrator practice differs from that of any other district owing primarily to two unique characteristics of the ore: (1) Native copper is present in all sizes from masses weighing many tons to microscopic size, which resists reduction in size when once liberated except by abrasion, and (2) the particles of native copper are tougher than the ore and hence are more resistant to sliming.

Careful analysis of these two properties of the ore accounts for much of what has been considered by some metallurgists as inefficient metallurgical practice and explains why the steam stamp has reigned supreme in the Lake copper district for 50 years in spite of an occasional attempt to replace it, although it has made no headway in other camps. It explains why stage crushing has not proved successful against single reduction from 10-inch through 3/16-inch mesh at one operation and the prevalence of jigs and roughing tables in spite of their recent abandonment in favor of all-flotation in many camps. A piece of copper the size of the head, the fist, or even of a pea presents rather a firm argument against either graded crushing or all-sliming practice.

Stamps were used on Lake Superior in the sixties, as soon as the mines developed any tonnage. Mechanically, the efficiency was low and was so recognized. The compound stamp (high- and low-pressure cylinders), developed about 1900, was a considerable improvement on the simple stamp; but modern practice is to run the stamp, simple or compound, noncondensing, and to use the exhaust steam in low-pressure turbines, thereby obtaining an excellent mechanical efficiency from the combined unit.

The only serious effort to displace the stamp as a primary crusher was made some 20 years ago by the Copper Range Co., which attempted stage crushing by means of rolls with intermediate jigging. It gave no promise and was soon abandoned. A Symons cone crusher might be able to master the nuggets of copper that clog other types of crushers or rolls, but existing mines are fully equipped, and with low stamping costs there is little inducement to warrant the capital expenditure necessary for a change in method.

Current practice calls for hand-picking of large nuggets from the rock, preceding stamping. When a large piece of copper escapes the vigilance of the picker and goes into the mortar it does no harm. It remains there, abrading slowly, until such time as a shoe is changed, when the content of the mortar is shoveled out if necessary and treated in a specially designed jig. Standard practice up to about 1900 was to clean out the mortar periodically, as often as every 4 hours in the early period of rich yields from the Calumet conglomerate, and to wash out the large nuggets of copper. About 1900 two devices were invented to remove this copper from the mortar hydraulically. The Krause mortar discharge was a hydraulic discharge of the ordinary type large enough to remove 4-inch nuggets, the copper within the mortar working down against the water pressure and being

trapped off periodically. The Woodbury mortar jig accomplishes the same purpose by means of a pulsating current operating within the mortar at the stave line and is effective not only for large pieces of copper but for smaller nuggets down to the 3/16-inch size that will pass through the mortar screen. This class of material constitutes only 3 percent of the total copper recovered from the conglomerate but as high as 30 percent of that from rich amygdaloid.

Before the introduction of the Wilfley table in 1898 the recovery of copper from slimes (local term for minus 60-mesh material) was very inefficient. Fortunately the values in the slimes were low because of the nonfriable nature of the valuable mineral. Revolving tables were used on primary slimes only, as the recovery of fines was so low that fine grinding by rolls or Chilean mills was not economic. With the gradual displacement of the revolving tables by Wilfleys, finer grinding was introduced, first by means of Chilean mills and later by Hardinge pebble mills. These pebble mills are still the standard fine-grinding units for treating conglomerate, ball mills being much more costly to run than pebble mills because of excessive abrasion of iron due to the hard rock. Leaching of fine sands was begun in 1916 and flotation of slimes soon afterward, so that the practice has not changed fundamentally for the past 12 years, and there have been no notable additions to existing plants and facilities in that time. The metallurgical efficiency, however, has been improving from year to year, as the following table shows:

Year	Assays, percent of copper			Extraction, percent
	Concentrates	Tailings	Heads	
1900	61.11	0.733	3.423	77.92
1905	61.36	.667	3.203	79.94
1910	56.79	.450	1.955	77.46
1915	58.50	.463	1.950	76.72
1920	57.98	.179	1.934	91.01
1925	63.17	.143	2.246	93.83
1926	62.18	.129	2.266	94.49
1927	60.18	.117	2.187	94.81
1928	61.31	.114	2.260	95.12
1929	61.27	.124	2.223	94.62

Although flotation practice has been used since 1919 for the treatment of Lake Superior conglomerate slimes, as indicated by Benedict, the application of flotation to the treatment of amygdaloid ores is comparatively recent. In 1928 Fahrenwald^{15/} demonstrated by laboratory methods that a considerable part of the copper lost in tailings of tables treating the amygdaloid ores was free from gangue but was in a fine or flaky state. Subsequent laboratory tests indicated that some of this tailing loss could be recovered by flotation methods.

Koepel^{16/} describes the remodeling of the Champion mill to include finer grinding and flotation treatment following Fahrenwald's work and states that the addition of flotation methods to jig and table treatment at this plant has increased decidedly concentrator recovery and improved considerably the grade of concentrates produced. Closed-circuit grinding of table tailings in ball mills with the bleeding and table treatment of part of the classifier return sand was adopted in the Champion mill at the same time.

Hadsel Mill

Announcement has recently been made that a well-known firm has taken over the manufacture and selling rights to the Hadsel mill. This mill performs the dual functions of crushing and grinding by elevating coarse material, 10-inch size or larger, by means of buckets placed at the periphery of a slowly rotating wheel and dropping the material from a maximum height of about 16 feet on stationary steel breaker plates. The crushing and grinding operations take place in closed circuit with a classifier, as in present ball-mill grinding practice. The information available at this time with reference to the performance of the mill is confined to operations of the Beebe Gold Mining Co. at Georgetown, Calif.^{17/} The Hadsel mill at this plant is a 24-foot duplex machine operated at a speed of 2.66 r.p.m. by a 100-hp. motor. It has crushed and ground siliceous gold ore at the rate of 150 to 215 tons per day with power consumptions amounting to 60 to 70 hp., which averages 6.41 kw.-hr. per ton, not including classifying. Although the mill has handled run-of-mine material successfully, the performance as noted was obtained with feed 10 inches in maximum size. A typical screen analysis of the finished product showed 8 percent plus 80-mesh, 6.5 percent minus 80- plus 100-mesh, 21.0 percent minus 100- plus 200-mesh, and 64.5 percent minus 200-mesh. Wear of buckets and breaker plates has not been determined, although it is believed to be low compared to ball and liner consumption of ball mills. Although the possibilities of the Hadsel mill in grinding sulphide copper ores have not been determined, the operating cost of crushing and grinding given as \$0.195 per ton for a capacity ranging from 150 to 215 tons per day, is low compared to present crushing and grinding operating costs at concentrators of larger capacities, as given in table 4, page 35.

^{15/} Fahrenwald, A. W., Copper Milling Research in Michigan: Rept. of Investigations 2878, Bureau of Mines, 1928, 5 pp.

^{16/} Koepel, Edward, Flotation of Native Copper at Freda, Mich.: Eng. and Min. Jour., vol. 131, 1931, p. 509.

^{17/} Hattl, J. B., Gravity Milling and Flotation of Low-Grade Gold Ore: Eng. and Min. Jour., vol. 133, 1932, p. 630.

Trends in Copper Content of Concentrator Heads, Recoveries,
and Concentrating Costs, 1911 to 1930, Inclusive

Trends in the copper content of concentrator heads, concentrator recoveries, and concentrating costs are presented in tables 1 to 3, respectively, for 1911 to 1930, inclusive. In presenting these trends concentrators were selected which operated almost continuously.

Copper Content of Concentrator Heads

Table 1 indicates the trends in copper content of mill heads for three typical concentrators which treat sulphide ores and for one mill in the Lake Superior district. The trend at each of the first three concentrators is downward; at Utah Copper 1.51 percent of copper in 1911 compared to 0.97 percent in 1930; at Miami 2.48 percent in 1911 compared to 0.72 percent in 1930, and at Hayden 1.83 percent in 1911 compared to 1.25 percent in 1928. The depletion of higher-grade concentrating ores is not the only cause for this trend, as improved metallurgical results and decreased mining and concentrating costs permitted the treatment of material in 1930 which could not be classed as "ore" some time ago. The Miami Copper Co. furnishes an interesting example of the grade of concentrator heads in 1930 as compared to those in 1912. The table which follows compares data for these years, and it may be noted that the heads in 1930 contained less copper per ton than the tailings rejected in 1912 and incidentally that the percentage recovery on the low-grade material in 1930 was higher than that on the higher-grade material in 1912 with little sacrifice in grade of concentrate; the cost of concentration per ton of ore treated in 1930 was less than half that in 1912.

TABLE 1. - Trends in copper content of heads at typical concentrators

Year	Copper in heads, percent			
	Utah Copper Co. ^{1/}	Miami Copper Co. ^{1/}	Nevada Consolidated Copper Co., Hayden mill ^{1/}	Calumet & Hecla Consoli- dated Copper Co., con- glomerate mill ^{2/}
1911	1.51	2.48	1.83	1.96
1912	1.36	2.39	1.68	1.91
1913	1.25	2.30	1.72	1.81
1914	1.42	2.28	1.76	1.78
1915	1.43	2.17	1.67	1.95
1916	1.43	2.07	1.61	1.97
1917	1.34	2.03	1.64	1.91
1918	1.23	2.03	1.61	1.74
1919	1.26	2.04	1.81	1.73
1920	1.16	1.97	1.72	1.93
1921	1.16	1.76	1.65	2.07
1922	1.26	1.88	1.59	2.53
1923	1.12	1.70	1.57	2.42
1924	1.07	1.62	1.52	2.36
1925	1.02	1.29	1.50	2.25
1926	1.01	.98		2.27
1927	.98	.86		2.19
1928	.99	.85	1.25	2.26
1929	.99	.83		2.22
1930	.97	.72		2.55

^{1/} From annual reports.

^{2/} Supplied by C. N. Benedict

Comparison of heads, metallurgical results, and costs of milling
at the Miami concentrator in 1912 and 1930

	1912	1930
Copper in heads ----- percent	2.39	0.72
Copper in tailings ----- do	.763	.153
Copper in concentrates ----- do	37.02	34.13
Mill recovery ----- do	69.39	78.88
Milling cost per ton of ore	\$.659	\$.297

The copper content of heads at the conglomerate concentrator of the Calumet & Hecla, as given in table 1, are somewhat higher since 1919, and the trend is different from that indicated for the concentrators treating sulphide ores. C. H. Benedict in a letter to the author attributes this condition to the fact that the average depth of the working faces of the mine has not increased materially since 1911 and that for the last 10 years at least the mining of old backs and shaft pillars in the upper, richer levels of the mine has accounted for a decided increase in the tenor of the ore concentrated. Benedict also states that selective mining is practical to the extent that stoping ground is chosen carefully and that the average width of the vein mined has decreased materially in recent years.

Trends in Concentrator Recoveries

The trends in recoveries of copper at typical plants are indicated in table 2 for 1911 to 1930, inclusive. At the beginning of this period recoveries at the three concentrators which treated sulphide ores ranged from about 63 to 74 percent; little, if any, improvement in recoveries is indicated in the data presented until the introduction of flotation. Since that time recoveries of sulphide copper have improved steadily, and although the first marked improvement was due to the treatment of slimes by flotation the later improvements due to finer grinding and more efficient classification as previously noted, had an important effect on recoveries.

The trend in recovery as shown in table 2 for the Miami Copper Co. is somewhat obscured by two conditions. First, the variable oxide copper content in heads from year to year appreciably affects recovery of total copper. For example, the drop in recovery in 1923 is explained¹⁸ by the unusually high-oxide copper content of the heads for this period. The second condition which affected recoveries at the Miami plant since 1925 as indicated in table 2, was the change which resulted from the working out of the "high-grade" ore body and the beginning of concentrating operations on the "low-grade" ore body. The treatment of the "low-grade" ore body necessitated radical changes in concentrator capacity and process due to its lower copper content and to an increase

¹⁸/ Annual Report of Miami Copper Co., 1928, p. 4.

in the proportion of pyrite to copper in this material. These conditions and an increase in the relative amount of oxide to sulphide copper account for decreases in recoveries at Miami since 1925. An examination of the sulphide copper content of Miami tailing, however, from 1913 to 1931 shows a gradual decrease in copper from 0.455 percent in 1913 to 0.060 percent for April 1931.

Referring to the improvements in recoveries made at the Conglomerate concentrator of the Calumet & Hecla Consolidated Copper Co., as indicated in table 2, Benedict, as previously noted, states that finer grinding by Chilean mills and later by pebble mills combined with the leaching of sands, which started in 1918, and the flotation of slimes in 1919 were mainly responsible for the improvement in recovery from 73 percent in 1911 to 91 percent in 1920.

TABLE 2. - Trends in recoveries of copper at typical concentrators

Year	Recovery of copper, percent			
	Utah Copper Co. ^{1/}	Miami Copper Co. ^{1/}	Nevada Consoli- dated Copper Co., Hayden mill ^{1/}	Calumet & Hecla Con- solidated Copper Co., conglomerate mill ^{2/}
1911	69.53	73.37	63.01	78.16
1912	66.32	69.39	68.28	78.31
1913	63.95	71.06	66.09	77.24
1914	66.04	69.93	67.83	75.90
1915	64.13	75.17	64.11	76.72
1916	62.34	73.83	70.20	76.60
1917	61.10	69.25	74.53	75.91
1918	65.11	71.15	74.92	80.75
1919	78.46	82.33	79.45	85.75
1920	81.38	82.43	81.98	91.01
1921	83.87	85.37	77.82	91.59
1922	80.00	86.36	76.66	93.30
1923	80.96	83.98	74.87	93.49
1924	85.94	80.17	78.27	93.00
1925	89.96	72.60	84.45	93.83
1926	87.01	76.77		94.49
1927	89.12	77.64		94.81
1928	85.56	68.41	87.06	95.12
1929	85.67	72.80		94.62
1930	89.00	78.83		95.70

^{1/} From annual reports.

^{2/} Supplied by C. H. Benedict.

Losses of Oxide Copper in Flotation Tailing. - Although rapid advances have been made in recent years in reducing the tailing loss of sulphide copper, very little has been accomplished in improving the recovery of oxide copper by flotation methods. Recoveries of oxide copper must, of course, be interpreted broadly owing to the difficulty of accurately determining the sulphide and oxide copper contents in heads and tailings by chemical methods. Present recoveries of oxide copper contained in feeds to flotation plants range from about 15 to 60 percent; most concentrators report a recovery less than 40 percent for 1929 and 1930. Past attempts to recover oxide copper by the use of a sulphidizing reagent have generally been effective. The Inspiration Consolidated Copper Co. solved its problem of mixed sulphide and oxide ore by a leaching method using sulphuric acid and ferric sulphate as solvents and electrolysis for recovery of copper from solutions. The dual process consisting of crushing to about 1/4-inch size and leaching with sulphuric acid for the recovery of oxide copper followed by grinding the leached residue and floating for the recovery of the sulphide copper was also considered and tested at Inspiration but was rejected in favor of direct leaching. A process such as is used at Inspiration, however, is not applicable to flotation tailings or to an ore in which the copper sulphide occurs as chalcopyrite; furthermore, the initial plant cost for such a process requires ore bodies sufficiently large to justify the expenditure. It is understood that experimental work is being carried on at two Southwestern concentrators to recover both the sulphide and oxide copper by flotation methods. The oxide copper is dissolved with a suitable reagent, and the copper is precipitated with sponge iron as cement copper which is subsequently recovered by flotation. This general plan was tried experimentally some years ago at one of the Southwestern plants but was not adopted then due to mechanical difficulties and to difficulties encountered in the floating of cement copper.

Trends in the Cost of Concentrating

Concentrating costs at typical plants in dollars per ton of ore treated from 1911 to 1930, inclusive, are indicated in table 3. The installed normal capacity in 1929 is given for each mill.

The object in presenting these data is not to compare the costs at different plants as such a comparison would be of little value unless the unit costs for power, labor, and supplies and operating conditions at each plant were given in detail. The figures are presented to show cost trends at each concentrator over a period of years and to indicate the effect of plant capacity on operating costs.

In general, concentrating costs per ton of ore treated decreased from 1911 to about 1914. From 1914 costs increased and reached a peak in 1919 or 1920. The latter period marked the introduction of flotation methods involving the cost of reagents and a constantly higher cost due to finer grinding together with abnormal conditions during and immediately following the war. Since 1919 to 1920 costs have shown a downward trend reaching a minimum during 1929 to 1931. It is interesting to note that for each of the first three plants listed in

table 3 the low point of the downward trend, reached between 1929 and 1931 was well below the concentrating cost per ton of ore in 1911. The increased concentrating cost at Calumet & Hecla in 1930 as compared to that in 1911 is due to the fact that the 1930 cost includes additional treatment of gravity tailing by regrinding, tabling, separation of sands and slimes, leaching of sands and flotation of slimes.

Besides the usual variations in costs of labor, power, and supplies over a considerable period of years certain changes have taken place in the concentration of copper ores since 1911 which have influenced the cost per ton of ore treated. Of these the most important is the general increase in plant capacities gained partly by the construction of additional concentrator sections to existing plants, but chiefly by a marked increase in the capacities of separate units or sections owing to advances in methods and equipment which have increased the efficiencies of crushing, grinding, flotation, and dewatering operations. A few comparisons of installed plant capacities in 1910 and 1911 with those in 1930 follow. The combined normal rated capacity of the Utah Copper Co. mills has increased from 9,000 tons per day in 1910 to 50,000 tons per day in 1929. At Miami, the normal rated capacity has increased from 2,500 tons per day in 1911 to 17,250 tons per day in 1930. At Hayden available capacity has increased from about 7,000 tons per day in 1913 to 12,000 tons per day in 1928. These capacities are for size of grinding that produces normal metallurgical results. With coarser grinding and corresponding sacrifice in recovery, Utah has operated at a peak capacity of about 69,000 tons per day and Hayden has capacity to treat about 16,000 tons per day. Hunt¹⁹ in describing the development of concentrating methods with special reference to increase in concentrator section capacity at Miami states:

The period of operation of the Miami concentrator, 1911 to 1930, has been one of great progress in methods of treating low-grade copper ores. During this period the company has seen the passing of the small-capacity mill operated by gravity-concentration methods and the gradual development of a process embodying all-flotation methods with greatly improved crushing, grinding, classifying, and flotation equipment. It is of interest to note that the capacity of the Miami plant has been increased from 2,500 to 17,250 tons per day and the increase of capacity has not only been accommodated in the original building but at present only about 65 percent of the original floor space is in use.

At the New Cornelia concentrator the capacity of each section has been increased from about 1,200 tons per day to approximately 1,375 tons per day, with finer grinding at present than was obtained under the old conditions.

¹⁹ Hunt, H. D., Milling Methods and Costs at the Concentrator of the Miami Copper Co., Miami, Ariz.: Inf. Circ. 6573, Bureau of Mines, 1932, p. 18.

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TABLE 3. - Trends in cost of concentrating copper ores^a

	Rated capacity, 1929; tons per 24 hours	Dollars per ton of ore treated									
		1911	1912	1913	1914	1915	1916	1917	1918	1919	1920
Utah Copper Co. ^{1/}	50,000	0.417	0.416	0.368	0.354	0.340	0.378	0.693	0.928	1.206	1.247
Miami Copper Co. ^{1/}	17,250	.627	.659	.572	.500	.579	.589	.691	.722	.924	.827
Nevada Consolidated Copper Co., Hayden mill ^{2/}	12,000	.595	.469	.519	.499	.509	.554	.829	1.010	1.290	1.240
Calumet & Hecla Consolidated Copper Co., conglomerate mill ^{3/}	3,800	.330	.327	.379	.348	.333	.375	.490	.766	.932	1.01
Britannia Mining & Smelting Co. ^{4/}	6,500										
Copper Queen, Phelps Dodge Corp. ^{5/}	4,300										
Nacozari, Phelps Dodge Corp. ^{6/}	2,500										
Engels Copper Co. ^{1/}	1,500										1.241

^a See page 17 for footnotes.

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TABLE 3. - Trends in cost of concentrating copper ores--Continued

Rated capacity, 1929; tons per 24 hours	Dollars per ton of ore treated											
	1921	1922	1923	1924	1925	1926	1927	1928	1929	1930	1931	
Utah Copper Co. ^{1/}	1.168	0.842	0.612	0.599	0.540	0.483	0.452	0.369	0.366	0.461	0.393	
Miami Copper Co. ^{1/}	.705	.644	.621	.637	.572	.491	.413	.382	.359	.297	.277	
Nevada Consolidated Copper Co., Hayden mill ^{2/}	1.220	.830	.640	.600	.630	.543	.469	.399	.395	.382		
Calumet & Hecla Consolidated Copper Co., conglomerate mill ^{3/}	.949	.729	.734	.738	.663	.662	.576	.533	.533	.495		
Britannia Mining & Smelting Co. ^{4/}			.710	.565	.517	.463	.430	.395	.378	.303	3/231	
Copper Queen, Phelps Dodge Corp. ^{5/}			.844	.703	.653	.568	.597	.531	.493			
Macozari, Phelps Dodge Corp. ^{6/}		.952	.818	.763	.600	.564	.504	.470	.555			
Engels Copper Co. ^{7/}	1.033	.931	.831	.849	.746	.678	.676	.678	.653	.757		

^{1/} From annual reports.

^{2/} The mines Handbook, vol. 18, pt. 1, p. 1533.

^{3/} Supplied by C. H. Benedict.

^{4/} Inf. Circ. 6619, Bureau of Mines, 1932, p. 6.

^{5/} Inf. Circ. 6404, Bureau of Mines, 1931, p. 6.

^{6/} Inf. Circ. 6358, Bureau of Mines, 1930, p. 5.

^{7/} Inf. Circ. 6550, Bureau of Mines, 1932, p. 4.

^{8/} 7 months.

The general increase in section capacity has taken place mainly since 1924 and has been accomplished largely by (1) introduction of the cone crusher; (2) finer and more uniform size feed to the grinding mills, larger size grinding and classifying units, substitution of balls for rods, and at some plants increasing the speeds of ball and rod mills; (3) introduction of the lime-xanthate-pine oil circuit in flotation, which increased the speed of flotation operations; depression of pyrite which decreased the amount of material to be floated; (4) dropping of pyrite and the use of lime, both of which have contributed to increased dewatering capacity per ton of mill feed, the former by increasing concentration ratios and the latter by producing faster settling pulps and pulps which were more readily filtered.

The direct effect of increased section capacities has been a marked increase in tons of ore treated per man-shift with a corresponding reduction in operation labor, repair labor, and overhead costs per ton of ore concentrated.

Power consumption expressed in kilowatt-hours per ton of ore treated has on the whole increased somewhat in grinding operations due to the much finer grinding at present. In flotation, however, the power consumption expressed in kilowatt-hours per ton of ore treated has on the whole decreased due to the faster floating conditions in present practice and at many plants to the use of matless or revolving-mat flotation cells. During the past few years there has been a marked reduction in unit power costs due to cheaper fuel and increased power-plant efficiencies; this reduction has amounted to as high as 40 percent. The combined effect of these factors on the cost of power per ton of ore treated has been a marked reduction since 1929 or 1930.

Simplification of flow sheets has assisted the reduction of concentrating costs over a long period. Decrease in the number of crushing stages by larger reduction ratios and elimination of gravity methods in concentration have reduced the number of machines in flow sheets. However, the tendency towards simplification of flow sheets, except in crushing and dewatering operations, has apparently been reversed at some plants in the past few years. In grinding operations the trend at some remodeled plants has been towards an increased number of grinding and classifying stages; this trend also is evident in some flotation operations, especially the number of roughing stages. The grinding of middlings and concentrates and the re-treatment of the ground products tend toward more complicated flow sheets, although the tonnage is relatively small as compared to the total concentrator feed. The separation of primary slimes in the concentrator feed and their separate treatment at some plants tends toward complexity of treatment. Any complication of process in late years has, of course, resulted either in improved metallurgical results or savings in power or supplies per ton of ore treated and is not comparable to the complex flow sheets of the days before flotation when neither simplicity of flow sheet nor high metallurgical efficiencies were realized.

PRESENT METHODS OF CONCENTRATING COPPER ORES

Crushing

The outstanding development in modern crushing practice is the success of the cone crusher and its general adoption in concentrator flow sheets. A tabulation of 23 crushing plants showed that **cone** crushers have been installed at 16 of them; furthermore, this machine has been installed in all recently constructed, large-tonnage crushing plants handling copper ores. A comparison of crushing plants operating with the cone crusher and those operating without it shows that the average size of the finished crushed product is less in the former and that the number of crushing stages has not been increased in obtaining the finer product. At many plants the size of finished product is finer although the number of crushing stages has been reduced by the introduction of the cone crusher.

A further study of plants indicates that crushing practice is rapidly becoming standardized in that ore at present is usually crushed to about 1/4 inch maximum size in three stages, irrespective of the maximum size of feed to the crushing plant. In the standardized practice the first stage comprises the usual gyratory or Blake-type crushers preceded by a grizzly to by-pass the fines; the second reduction is effected by a cone crusher and the third by rolls, the latter usually operating in closed circuit with vibrating screens.

The advantages gained at the New Cornelia crushing plant of the Phelps Dodge Corporation and the crushing plants of the Utah Copper Co. and Cananea Consolidated Copper Co. by the installation of the cone crusher are typical; descriptions of changes in crushing operations at these plants follow.

The original New Cornelia concentrator of five sections had capacity to treat 1,200 tons of ore per section until 1928, when changes in crushing, grinding, and flotation practices increased the capacity to 1,875 tons per section.^{20/} Important changes in the crushing department added approximately 240 tons per day to the capacity of each concentrator section or about 35 percent of the total increase in section capacity since 1928. In 1928 crushing New Cornelia open-pit hard ore to approximately 1/2-inch size was done in four stages, as follows: Gyratory crusher to 6 inches, secondary gyratory crushers to 2.5 inches, disk crushers to 1.25 inches, and rolls operated in open circuits to 1/2 inch nominal size. In 1929 cone crushers were installed, replacing the secondary gyratory and disk crushers. The rolls were placed in closed circuit with vibrating screens having 1/4-inch openings. A comparison of 1928 and 1932 methods and results follows.

^{20/} Supplied by M. Curley, manager, New Cornelia Branch, Phelps Dodge Corporation.

Performances of the New Cornelia crushing plants of 1928 and 1932

	1928	1932
Number of crushing stages	4	3
Last crushing stage in closed circuit with screens	No	Yes
Size of finished product:		
Plus 4-mesh ---- percent	56.0	14.9
Minus 65-mesh . . . do	9.0	15.7
Capacity per unit, tons per hour:		
Maximum	137	275
Average	121	202
Power consumed ---- kw.-hr. per ton	1.93	2.76
Labor, tons per man-shift:		
Operation	104.	117
Repairs	307	349
Total	78	88
Cost of supplies per ton of ore crushed	\$0.070	\$0.057

In summarizing changes in New Cornelia crushing practice, Curley states that installing four 7-foot cone crushers and operating the rolls in closed circuit with vibrating screens reduced the size of the crushing plant product as indicated in the preceding tabulation, which in turn increased the capacity of the grinding units by 20 percent.

Martin,^{21/} in describing the changes resulting from the adoption of the cone crusher in the Arthur and Magna plants of the Utah Copper Co., states:

Up to 1929 the no. 27 gyratory crusher product of both plants passed through no. 7 1/2, no. 8, or no. 9 gyratory crushers, where it was reduced from 6-inch to 3-inch maximum size, then through two stages of 74- by 20- or 54- by 20-inch Garfield rolls in closed circuit with screening towers, the screen undersize product containing approximately the same sizes as the Symons crusher product produced at present, (i.e., about 4 percent plus 1 inch). In 1929 four 7-foot Symons cone crushers were installed at each plant, and these crushers reduce the 6-inch material in one pass to 3/4 to 1 inch size, thus eliminating 9 small gyratory crushers and 14 sets of large rolls.

^{21/} Martin, H. S., Milling Methods and Costs at the Arthur and Magna Concentrators of the Utah Copper Co.: Inf. Circ. 6479, Bureau of Mines, 1931, p. 6.

The advantages gained at these mills by installing cone crushers, according to Martin's description, are: (1) simplification of the crushing flow sheet by a 6 to 1 reduction ratio in cone crushers, this reduction formerly requiring three stages; (2) installation of the larger-capacity cone-crusher units which eliminated many small-capacity crushing units formerly used. Cost of crushing to approximately 1-inch size in 1925 at Utah Copper Co. mills having an average daily capacity of about 34,000 tons, before the installation of the cone crushers, is given by Taggart²² as \$0.0390 per ton; in 1929, after the installation of cone crushers, the cost was \$0.0197 per ton according to Martin for 1 month's operation with a daily capacity of about 43,500 tons.

A. T. Tye, assistant concentrator superintendent of the Cananea Consolidated Copper Co., supplied the following information concerning the remodeling of the crushing plant at Cananea in 1931. Formerly a reduction from a maximum size of 15 inches to a nominal size of 1 1/2 inches was effected in two stages; first the ore was crushed to 3 1/2 inches in a gyratory, then it was further reduced to 1 1/2 inches by two disk crushers. A shaking screen with 1-inch openings preceded the disk crushers. The plant as remodeled consists of two identical units each equipped with a grizzly having 4-inch spaces, one no. 8 gyratory crusher, one 4- by 7-foot vibrating screen with 3/4-inch openings, and one 5 1/2-foot cone crusher. The vibrating-screen undersize and cone-crusher product, 5/8- to 1-inch nominal maximum size, comprise the feed to the grinding mills. A comparison of the operating results of these two plants follows.

Rolls

The use of rolls in the final crushing stage is almost general in current practice at copper concentrators. A study of the 23 crushing plants in the United States, Canada, and Mexico previously mentioned showed that rolls are used in 13 for the final crushing operation and, of the latter, 16 operate the last roll stage in closed circuit with screens, thus providing a uniform maximum size of feed to the grinding units.

Modern roll practice favors large machines, 72 to 80 inches in diameter, operated at high speeds under choke crushing conditions with heavy circulating loads (as much as 500 to 600 percent of initial feed) between the rolls and the screens which follow them.

Costs of coarse and intermediate crushing are given in table 4. Table 6 includes power consumed in coarse and fine crushing and table 8 labor for typical crushing plants.

Grinding and Classifying

A summary of 25 copper-concentrator grinding flow sheets in the United States, Canada, and Mexico showed that single-stage grinding circuits were used at 9, double-stage at 15, and three-stage at 1.

²²/ Taggart, A. F., *Handbook of Ore Dressing*: John Wiley & Sons, Inc., New York, 1927, p. 90.

Comparison of performances of old and new crushing plants at Cananea

	Old crushing plant, 1 gyratory and 2 disk crushers, first half of 1929	New crushing plant, 2 gyratory and 2 cone crushers, last half of 1931
Number of crushing stages	2	2
Number of screening stages	1	2
Maximum size of feed --- inches	15	15
Maximum nominal size of product do	1 1/2	3/4
Capacity --- tons per hour	220	600
Power consumed for crushing and conveying ----- kw.-hr. per ton	1.051	1.129
Power consumed for grind- ing and classifying do	7.658	6.801
Power consumed for crush- ing and grinding do	8.709	7.930
Tons crushed per man-shift	100	343
Cost of crushing and conveying, per ton	\$0.0587	\$0.0244
Cost of grinding and classifying per ton	.2141	<u>1/</u> .1617
Cost of crushing and grinding per ton	.2728	<u>1/</u> .1861

1/ In addition to providing finer feed to grinding units, the speed of ~~rod~~ mills was increased from 16.5 to 18.5 r.p.m., thereby increasing capacity of rod mills by 10 percent and accounting partly for decreased grinding cost.

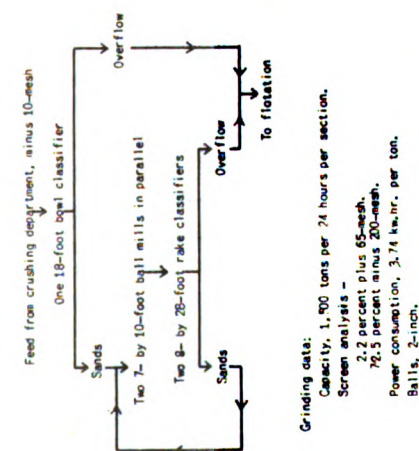


Figure 1.—Flow sheet of one of seven grinding sections at the Hayden plant of the Nevada Consolidated Copper Co. (after Gamis, W. I., Concentrator Methods and Costs at the Hayden Plant of the Nevada Consolidated Copper Co.; Inf. Circ. 624; Bureau of Mines, 1930, 25 pp.)

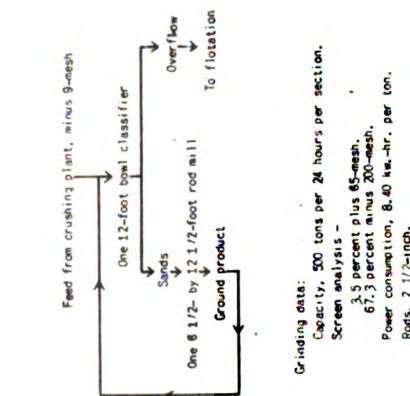


Figure 2.—Flow sheet of one of sixteen grinding sections at the Phelps Dodge Concentrator, Bisbee, Ariz. (after Macdonald, W. J., Selective Flotation Mill at Copper Cliff; Eng. and Min. Jour., vol. 130, Nov. 10, 1930, p. 470.)

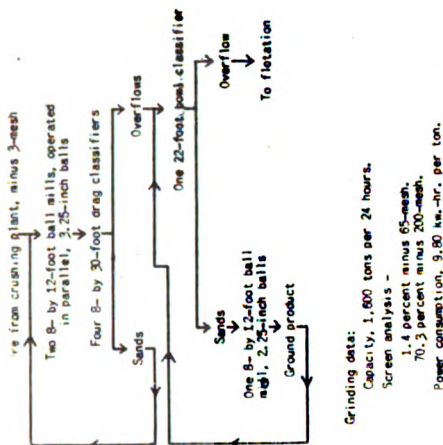


Figure 3.—Flow sheet of grinding used at the United Verde mill. (after Kuzall, C. R., and Barker, L. W., Milling Methods and Costs at the Concentrator of the United Verde Copper Co., Clankdale, Ariz.; Inf. Circ. 6343, Bureau of Mines, 1930, 28 pp.)

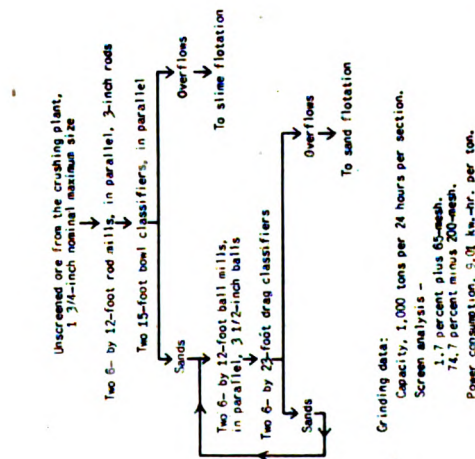


Figure 4.—Flow sheet of one of four grinding sections at the Copper Queen mill. (after Wittenau, E., and Cramer, W. R., Milling Methods and Costs at the Copper Queen Concentrator of the Phelps Dodge Corporation, Bisbee, Ariz.; Inf. Circ. 6404, Bureau of Mines, 1931, 29 pp.)

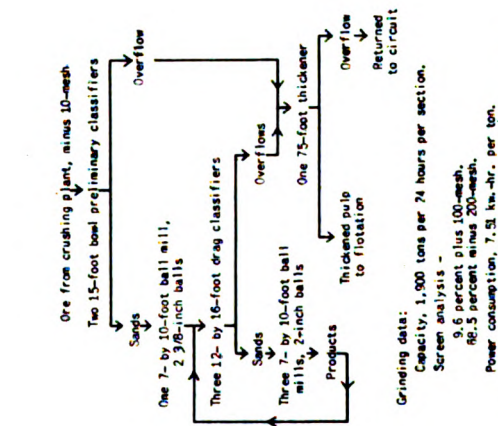


Figure 5.—Flow sheet of one of thirteen grinding sections, Arthur mill of the Utah Copper Co. (after Martin, H. S., Milling Methods and Costs at the Arthur and Magna Concentrators of the Utah Copper Co.; Inf. Circ. 6479, Bureau of Mines, 1931, 25 pp.)

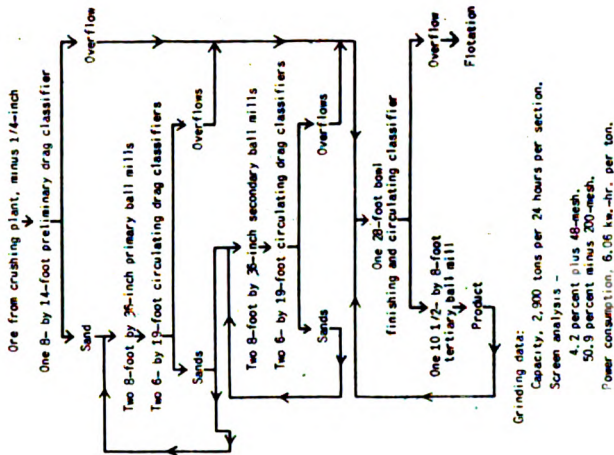


Figure 6.—Flow sheet, one of six grinding sections at the Miami mill. (after Hunt, H. D., Milling Methods and Costs at the Concentrator of the Miami Copper Co., Miami, Ariz.; Inf. Circ. 6573, Bureau of Mines, 1932, 25 pp.)

Single-stage Grinding Circuits

At 9 concentrators using single-stage grinding circuits, 6 used ball mills, 2 operated rod mills, and 1 (the Calumet & Hecla conglomerate mill) retained pebble mills due to the hardness of the material handled.

Typical examples of single-stage circuits are given in figures 1 and 2. At the Hayden plant of the Nevada Consolidated Copper Co. (fig. 1) the product from the crushing plant is received by a preliminary bowl classifier which delivers sands to the ball mills and overflow to flotation. The ball mills are operated in closed circuit with single-stage rake classifiers which return the sands to the ball mills and deliver overflows to flotation. The crushing plant product contains about 52 percent minus 100-mesh; the preliminary bowl classifier removes about 45 percent of the new ore feed prior to grinding.

The flow sheet of 1 of 16 grinding sections at the concentrator of the International Nickel Co. is given in figure 2. In each section a bowl classifier operates in closed circuit with a rod mill. The crushing-plant product is received by the classifier; fines are therefore removed prior to grinding. The crushing-plant product is minus 9-mesh and contains about 25 percent minus 100-mesh material.

Two-stage Grinding Circuits

In the primary circuits of 15 double-stage grinding circuits 11 contained ball mills and 4 operated with rod mills; in the secondary circuits 14 used ball mills and 1 retained the rod mill.

Typical examples of two-stage grinding circuits are given in figures 3 to 5, inclusive. At the United Verde concentrator (fig. 3) the crushing-plant product is fed to two primary ball mills which are operated in parallel and in closed circuit with rake classifiers. The combined primary classifier overflows are reclassified in a bowl classifier which operates in closed circuit with one secondary ball mill. The United Verde concentrator feed comprises schist and massive sulphide ore. The dissemination of the sulphides in the gangue of the schist is such that grinding to minus 65-mesh is sufficient to effect liberation, but the association of the chalcopryrite with pyrite and marmatite is very fine, especially in the massive sulphide ore. Kuzzell and Barker²³ describe the results of large-scale grinding tests on United Verde ores with single- and double-stage grinding circuits and state that relatively fine grinding of all the ore, as distinguished from **regrinding**, of concentrates (practiced at the Miami and Britannia concentrators) is essential, or at least that initial grinding should be carried much further than is customary where regrinding of the concentrate is done. They state further that two-stage grinding on United Verde ore is more efficient than single-stage grinding, as indicated in the comparison which follows.

²³/ Kuzzell, C. R., and Barker, L. M., Milling Methods and Costs at the Concentrator of the United Verde Copper Co., Clarkdale, Ariz.: Inf. Circ. 6343, Bureau of Mines, 1930, p. 12.

	Rate per 24 hours, tons	Minus 100-mesh product produced, tons	Minus 100-mesh product produced, kw.-hr. per ton
Single-stage	1,200	840	11.4
Two-stage	1,600	1,280	10.6

At the Copper Queen concentrator (fig. 4) the primary circuit consists of rod mills operated in open circuit; they receive unscreened feed of nominal 1 3/4-inch maximum size and deliver the ground products to bowl classifiers. The secondary circuit comprises ball mills operated in closed circuit with drag classifiers. Wittenau and Cramer,²⁴ in referring to the coarse feed handled by the grinding circuits, state that general practice favors a finer feed for the grinding mills but that unfortunately Copper Queen ore bodies lie in the drainage region of a considerable watershed. During the rainy season much difficulty is encountered in crushing even to the coarse sizes noted, and production would be practically nil if finer crushing were attempted.

Prior to 1923 both primary and secondary grinding at Copper Queen was done in rod mills. Wittenau and Cramer state that extensive testing of Copper Queen ores indicated that ball mills were considerably more efficient for closed-circuit-grinding. The secondary rod mills were therefore converted to ball mills, retaining the same speed as when operated with rods. Short-wave liners replaced long-wave liners, high-center discharge replaced low-center discharge, and 3 1/2-inch chilled cast-steel balls replaced 3-inch, high-carbon steel rods.

Copper Queen concentrating ore contains copper, mainly as chalcocite, which is very finely associated with pyrite in a siliceous gangue. The classifying and grinding arrangement used produces a segregation of primary slime in the overflow of the bowl classifier, which Wittenau and Cramer state is of decided advantage in allowing separate treatment of primary slime by flotation. Another advantage in the arrangement is that the chalcocite-pyrite combination segregated in the secondary grinding circuit allowing the latter to be used primarily in freeing chalcocite from pyrite.

Although the Copper Queen primary grinding circuits have been operating in open circuit, Wittenau and Cramer state that large-scale test work has indicated that the tonnage handled could be increased 10.0 percent by the use of primary circulating classifiers.

²⁴ Wittenau, K., and Cramer, W. B., Milling Methods and Costs at the Copper Queen Concentrator of the Phelps Dodge Corporation, Bisbee, Ariz.: Inf. Circ. 6404, Bureau of Mines, 1931, p. 8.

The use of a preliminary classifying stage in a two-stage grinding circuit is illustrated in the flow sheet of the Arthur plant of the Utah Copper Co. (fig. 5) which comprises two preliminary bowl classifiers producing overflows for thickening and flotation and sands for feed to one primary ball mill. The ground product of the primary mill is distributed to 3 drag classifiers which operate in closed circuit with 3 secondary ball mills.

Three-stage Grinding Circuits

The Miami concentrator is the only plant of the 25 considered that uses three stage initial grinding. Preliminary classifying (fig. 6) is followed by primary grinding with ball mills in partly closed circuit with circulating drag classifiers; secondary grinding is done with ball mills in closed circuit with circulating drag classifiers and tertiary grinding by ball mills in closed circuit with bowl classifiers. The bowl classifiers of the tertiary grinding circuit also act as finishing classifiers in that they receive the overflow products of all preceding classifiers and handle all the feed to the section, thus controlling the character of the final ground product delivered to the flotation machines. In addition to the three initial grinding stages described, grinding circuits also are provided for the regrinding of concentrates and middlings produced in the bulk flotation circuits.

Dorr and Marriott^{25/} in their paper describing grinding and classifying arrangements at 9 copper concentrators give many interesting comparisons showing recent progress at these plants in increasing the efficiency of grinding operations. In conclusion they present a suggested grinding circuit as follows:

It is evident from examining these sections (the nine flowsheets used to represent modern grinding practice) that no two properties have developed along quite the same lines and that no single flow sheet can be said to represent the optimum of present practice. This is inevitable since each concentrator has had its own particular ore-dressing problems, modified by existing plant conditions.

As a means of summarizing ***, however, a generalized flow sheet *** (is presented) which may be regarded as a composite picture of developments to date. *** Data exist to prove the value of each of the steps of this flow sheet at one or more concentrators.

Dorr and Marriott's suggested flow sheet comprises three grinding stages, the last designated as a mineral grind. The primary grinding stage is a closed-circuit operation. Five classifying stages are provided, including preliminary and intermediate classifiers and the three circulating classifiers of

^{25/} Dorr, J. V. N., and Marriott, A. D., Importance of Classification in Fine Grinding; Milling Methods, 1930; Am. Inst. of Min. and Met. Engrs., p. 143.

the three grinding stages. The last circulating classifier acts as a finishing classifier in addition to its closed-circuit function. The intermediate and finishing classifiers are of the double-stage bowl type, while the others are single-stage rake machines. Provisions are made to by-pass all fines in the crushing-plant product to the intermediate classifiers, to reclassify the overflows from the first two grinding stages in the intermediate classifier, and to classify the total feed in the finishing classifier.

The number of stages and arrangement of machines provides for (1) confining the range of reduction in each grinding stage; (2) removing and by-passing fine material, thus relieving the grinding circuits; (3) sufficient classifier capacity not only to relieve the grinding circuits but also to increase the efficiency of classification by two reclassifying operations, the last of which is designed to drop sulphides, notably pyrite associated with copper sulphides, for a so-called "mineral grind"; and (4) final control of flotation feed by one machine.

The flow sheet of grinding at the Miami mill (fig. 6) approaches the flow sheet suggested by Dorr and Marriott; in general, increase of classifier capacity has been the rule in remodeling grinding units at copper plants in recent years, preliminary, intermediate, and finishing classifiers having been installed in many concentrators to relieve the grinding circuits of fine material or to increase the efficiency of classification. On the other hand, recently constructed concentrators, including International Nickel, Flin Flon, Sherritt Gordon, and Noranda, have relatively simple grinding flow sheets in that three have one grinding stage and the fourth has two grinding stages; three have one classifying stage, while the fourth has two classifying stages to provide aeration between grinding and flotation operations.

Table 4 gives costs of grinding and classifying operations at copper concentrators, table 6 power consumed for grinding and classifying, and table 8 labor for typical grinding sections.

Flotation and Dewatering

In the summary of flotation operations at copper concentrators 24 of the larger plants were considered, of which 16 were in the United States, 2 in Mexico, 5 in Canada, and 1 in Cuba. The total tonnage handled by the 16 concentrators in the United States accounted for 90 percent of the total copper ore concentrated by flotation in the United States in 1929. The total installed normal daily capacity of the concentrators considered was 180,900 tons.

Flotation Machines

Of the total installed capacity considered (180,900 tons) 68,450 tons per day was for all types of mechanical-agitation machines and 112,450 tons for all types of pneumatic machines. The mechanical-agitation flotation equipment included straight mechanical agitation, subaeration, and mechanical air

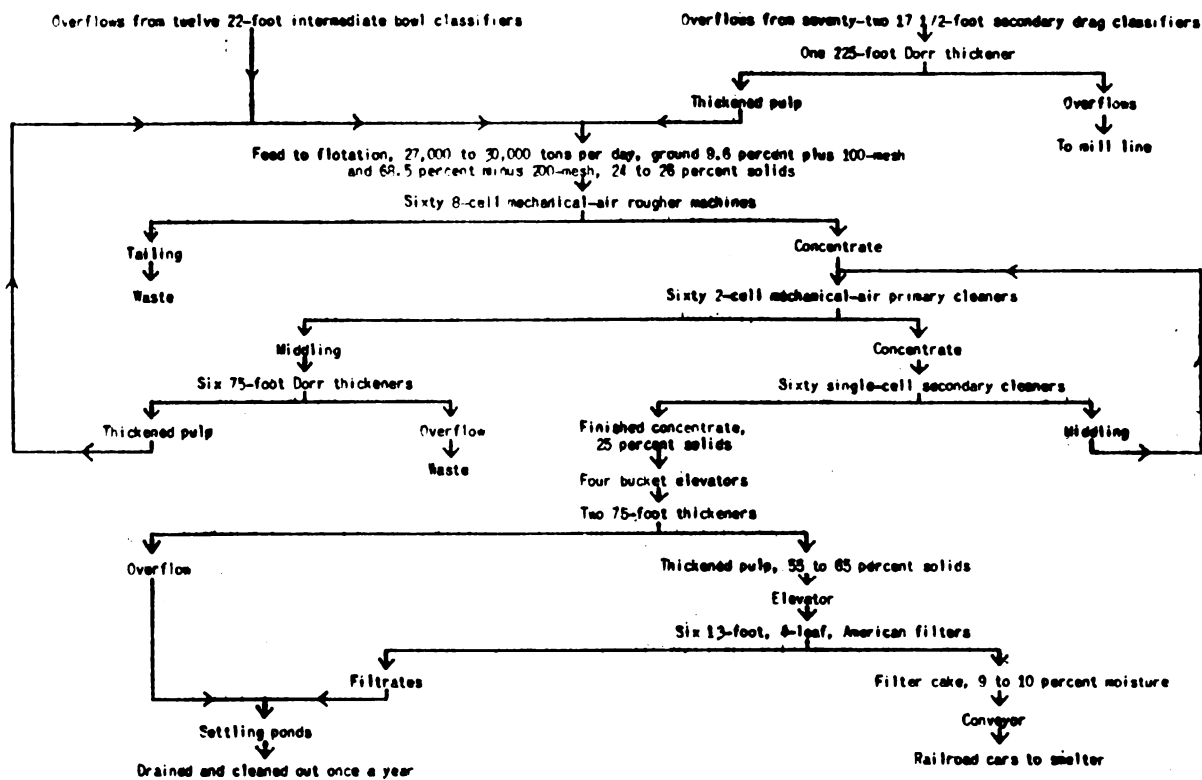


Figure 7.—Flow sheet of flotation and dewatering at the Magma mill of the Utah Copper Co. (After Martin, H. S., *Milling Methods and Costs at the Arthur and Magma Concentrators of the Utah Copper Co.*; Inf. Circ. 6479, Bureau of Mines, 1931, 25 pp.)

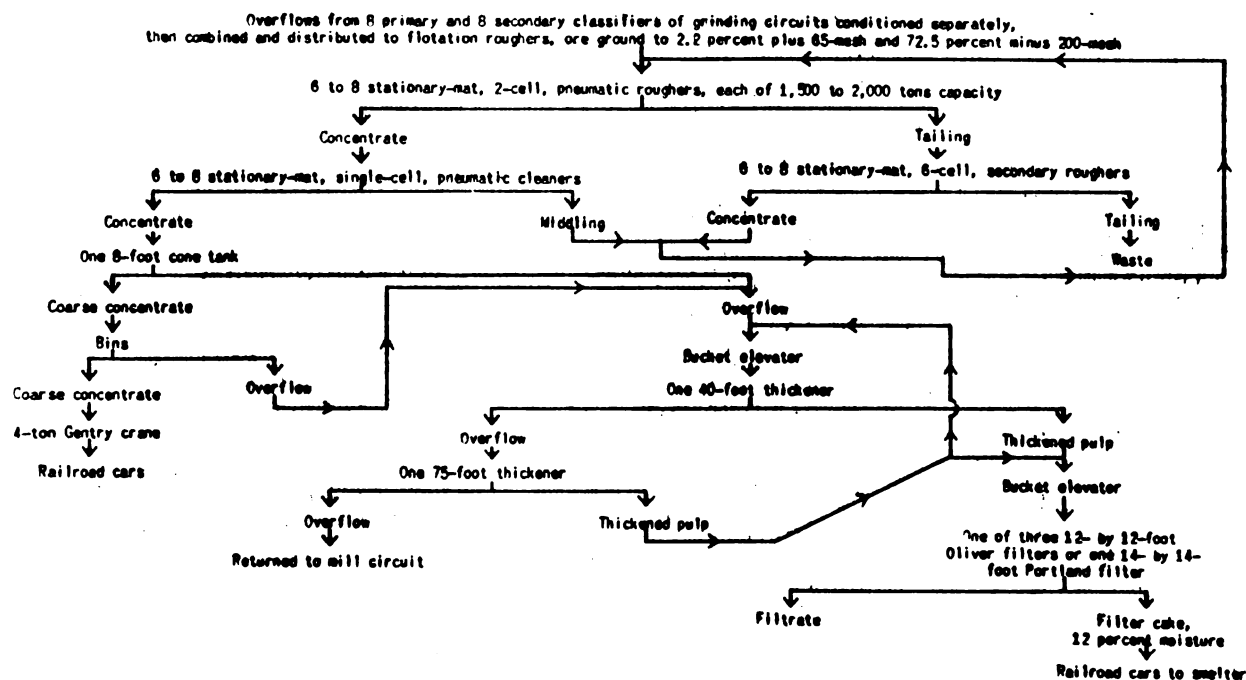


Figure 8.—Flow sheet of flotation and dewatering at the Hayden mill of the Nevada Consolidated Copper Co. (After Gerns, W. I., *Concentrator Methods and Costs at the Hayden Plant of the Nevada Consolidated Copper Co.*; Inf. Circ. 6241, Bureau of Mines, 1930, 25 pp.)

machines. The pneumatic cells comprised stationary-mat cells of the Callow type, rotating-mat cells of the McIntosh type, and matless cells such as Forrester, Miami, and Southwestern machines. The Magna and Arthur concentrators of the Utah Copper Co. accounted for 50,000 tons of the 68,450 tons treated by the mechanical-agitation machines, leaving only 18,450 tons for all other concentrators using this type of machine.

Mechanical-agitation machines.— The mechanical-agitation machine is used at the Magna and Arthur concentrators of the Utah Copper Co., at the Flin Flon, Sherriitt-Gordon, and United Verde concentrators, at the Anaconda plant for the treatment of ground sands after removal of slimes, and for primary flotation at the Calumet & Hecla and Engels concentrators.

The flow sheet of flotation treatment at the Magna concentrator is given in figure 7; it comprises one stage of roughing in 8-cell mechanical-air machines and cleaning of rougher concentrate in two stages. Overflow pulps from the secondary classifiers of the grinding units are thickened before joining the primary classifier overflow pulps for flotation. Middling pulps from the cleaner flotation machines are also thickened before being returned to the head of the rougher circuits. The power consumption for flotation at the Utah Copper Co. plants for April 1930 amounted to 4.92 kw.-hr. per ton of ore treated compared to an average of 1.77 kw.-hr. per ton at concentrators using pneumatic-type machines. Martin^{26/} in commenting upon the retention of mechanical-air cells at the Utah Copper Co. mills states that an experimental flotation plant is maintained at the Arthur concentrator where many types of flotation machines, including a 400-ton unit of the matless pneumatic type, have been tested on full plant scale and that the results of these tests, using machines more economical in power than the mechanical-air type employed at Magna and Arthur, have not shown any over-all saving from the standpoints of metallurgical results and consumption of reagents.

Pneumatic machines.— The Callow-type machine accounted for 11.1 percent, the McIntosh for 14.4 percent, and all matless pneumatic cells for 36.6 percent of the total installed flotation capacity at the 24 concentrators considered.

The three types of matless cells used in current copper-concentrator practice - namely, the Forrester, Hunt-Miami, and Southwestern - are usually designated individually and accepted as distinct types, but it is understood that the patentees have pooled their interests and that all three types are now controlled by the same company.

Typical examples of pneumatic machine installations are given in figures 8 to 10, inclusive.

^{26/} Martin, H. S., Milling Methods and Costs at the Arthur and Magna Concentrators of the Utah Copper Co.: Inf. Circ. 6479, Bureau of Mines, 1931, 25 pp.

Flotation at the Hayden concentrator of the Nevada Consolidated Copper Co. is effected in two roughing stages employing the stationary-mat-type machine with single cleaning of primary rougher concentrate. (See fig. 8.) Secondary rougher concentrate and cleaner middling are returned to the head of the flotation circuit. Power consumption for flotation at Hayden, including pumping in the flotation circuit, amounted to 1.76 kw.-hr. per ton in 1928. Garms^{27/} states that stationary-mat type cells were retained at Hayden after very thorough testing with a matless machine of the multiple-pipe type as it was found that the former produced slightly better metallurgical results when treating Ray ores.

Figure 9 gives the flow sheet of the flotation circuits at the concentrator of the International Nickel Co. The separation of about one half of the copper in the form of chalcopyrite is obtained by selective flotation in the presence of pentlandite and pyrrhotite in two roughing stages with double cleaning of combined primary and secondary rougher concentrates. The finished copper concentrate contains approximately 25 percent of copper and 1.25 percent of nickel. MacDonald^{28/} states that the copper concentrate is floated in the presence of pentlandite and pyrrhotite by crowding the rougher machines and using minimum amounts of reagents none of which is directly intended to depress either pentlandite or pyrrhotite.

After the copper concentrate is removed, pentlandite and most of the remaining chalcopyrite are floated in three roughing stages and cleaned; this produces finished nickel-copper concentrate. The flotation tailing is passed over scavenger tables to recover all sulphides, especially pyrrhotite, as pyrrhotite cannot be rejected without serious losses of nickel and precious-metal values.

MacDonald, in commenting on the installation of the rotating-mat machine, states further:

The use of pneumatic flotation machines on such heavy sulphide ore as this would perhaps not have been considered feasible a few years ago; but it is believed that with the type of cell employed throughout this mill fully as good metallurgical results will be obtained as though mechanically agitated machines were used. The power requirement for these cells is low compared with that needed by mechanically agitated machines of equal capacity, and the mechanical upkeep also promises to be small. All cell rotors are driven at 16 r.p.m. by individual 2-horsepower motors through reduction gears. The cells are intentionally overpowered to provide for surge loads and to permit easy starting after forced shutdowns. The rotors are of punched sheet metal and are 9 inches in diameter. Perforated rubber blankets are used as the porous media covering the rotors,

^{27/} Garms, W. I., Concentrator Methods and Costs at the Hayden Plant of the Nevada Consolidated Copper Co., Ariz.: Inf. Circ. 6241, Bureau of Mines, 1930, p. 3.

^{28/} MacDonald, W. T., Selective Flotation Mill at Copper Cliff: Eng. and Min. Jour., vol. 130, Nov. 10, 1930, p. 470.

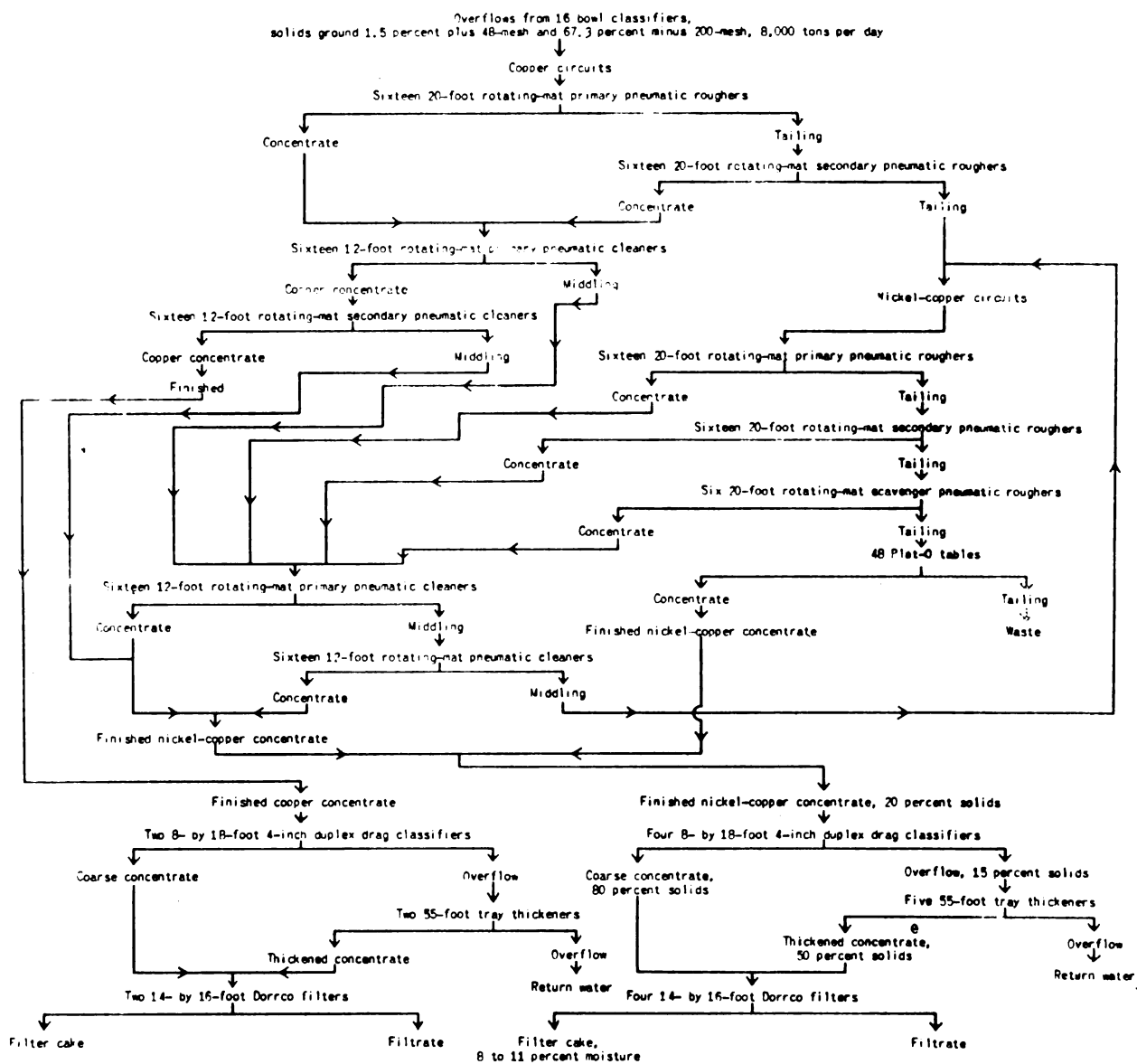
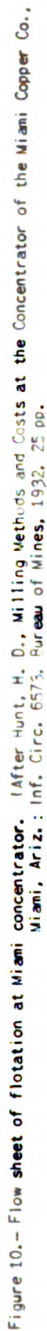


Figure 9.—Flow sheet of flotation at the concentrator of the International Nickel Co. (After MacDonald, W. T., *Selective Flotation Mill at Cooper Cliff*; Eng. and Min. Jour., vol. 130, Nov. 10, 1930, p. 485.)



the perforations of approximately 0.01-inch diameter numbering about 200 per square inch. Although operating in a lime circuit, no perceptible interference with proper air distribution, such as might result from surface incrustation or blinding of perforations, has developed. Flotation air at the blowers is at 2 1/4 pounds pressure, but the actual pressure at the cells averages slightly under 2 pounds, varying within narrow limits.

At Miami (fig. 10) bulk-flotation concentrate containing chalcocite and pyrite is produced in matless pneumatic cells. This concentrate is reground and treated in a selective flotation circuit for the separation of chalcocite and pyrite. Hunt,²⁹ in describing the control of flotation operations in the selective circuit at Miami, stresses the importance of aeration before flotation, and among the conditions he gives for successful results are the following:

Miami experience has shown that the rejection of most of the insoluble content after fine grinding is a simple matter but that the rejection of pyrite without considerable deadening of the chalcocite is somewhat difficult owing to the fact that in high-alkaline circuits pyrite and chalcocite have very similar flotation characteristics.

Lime is used to depress the pyrite, and careful control of alkalinity is necessary. The selective-flotation operation is effected in a pulp which contains water saturated with lime and, in addition, undissolved calcium oxide. It has been found, however, that if too great an excess of undissolved calcium oxide is present the selective action is not obtained.

Other depressants, such as potassium or sodium cyanide, have not proved effective for the treatment of Miami concentrate.

Thorough conditioning of the pulp is essential. This conditioning not only requires agitation but also thorough aeration of the pulp.

The conditioning at Miami is effected in three preagitator cells placed in series before the roughing operation in the selective circuits. The rough concentrate is aerated further, before cleaning, in four preagitator cells placed in series.

Total power consumption for flotation operations at Miami was 2.35 kw.-hr. per ton of ore treated in 1930.

Performance of flotation machines.— It is difficult to present convincing data on the comparative performance of various types of flotation machines that are fair to all machines unless the figures so presented are obtained at the same plant under comparable conditions; the data presented herein, therefore, should be considered with this in mind.

²⁹/ Hunt, H. D., Milling Methods and Costs at the Concentrator of the Miami Copper Co., Miami, Ariz.: Inf. Circ. 6573, Bureau of Mines, 1932, p. 11.

A study of the operating data from 24 concentrators treating copper ores with various types of flotation machines showed power consumptions as follows. At concentrators employing the mechanical-agitation machine power consumption ranged from 2.12 to 4.92 kw.-hr. per ton with an average of 4.75 kw.-hr. per ton, whereas at plants using one of the pneumatic-type cells power consumption ranged from 0.80 to 6.65 kw.-hr. per ton and averaged 1.77 kw.-hr. The average power consumptions for flotation at plants employing different-type pneumatic machines were 2.19, 1.31 and 1.81 kw.-hr. per ton of ore treated for stationary, rotating, and matless types of cells, respectively.

A further study of the data representing recent practice at copper concentrators indicates that the matless-type cells are handling flotation feeds which are coarser on the average than feeds to other types of pneumatic machines. The average minus 200-mesh material in flotation feeds to stationary-mat cells at 4 plants amounted to 67.7 percent compared to 60.0 percent at 5 plants employing rotating-mat cells and 54.0 percent at 6 concentrators using matless-type machines. The plus 65-mesh material amounted to 6.4 percent for stationary-mat, 4.6 percent for rotating-mat, and 12.9 percent for matless machines. The screen analysis of flotation feed at the Britannia concentrator for July 1931 indicates that exceptionally coarse feed is being handled by matless-type cells. At this plant sand feed to the bulk flotation circuit averaged 3 percent plus 28-mesh, 9 percent plus 35-mesh, 20 percent plus 48-mesh, 31.5 percent plus 65-mesh, and 39.5 percent minus 200-mesh in July 1931.

At Britannia the Forrester machines used as roughers in the bulk-flotation circuit are equipped with 1/2-inch diameter air pipes spaced 2 1/2 inches apart for the first 20 feet and 3 inches apart for the remaining distances, as compared to standard 3/4-inch air pipes spaced 4 inches apart as used by Forrester at the Old Dominion plant. At Britannia air consumption was 45 cubic feet of free air per minute per linear foot of machine (9,000 cubic feet per ton of mill feed) at 1.75 pounds pressure at the cells, and power for blower air was 1.24 kw.-hr. per ton of ore milled during July 1931.

A tabulation of flotation costs shows that the matless-type cell operates at a lower average cost than other types of flotation machines due to savings in power, labor, and supplies.

The trend of present practice indicates a preference for one of the newer types of pneumatic cells when the savings in cost are consistent with the metallurgical results obtained.

Control of Flotation Operations

The conditioning of the ore pulp with lime, control of reagent quantities, and the places at which reagents are added, form the basis of controlling flotation operations at concentrators that use selective-flotation methods for the depression of pyrite or pyrrhotite.

Lime. - The usual practice in the treatment of sulphide copper ores is to add lime to the grinding or crushing units. At a few concentrators where sand and slime are treated separately and where dilution of slime necessitates thickening before flotation the lime is added to the pulp after thickening as undissolved lime tends to segregate in classifier overflows. If the thickened slime is to be treated by selective methods and lime used for the depression of pyrite, agitating equipment for aeration and conditioning usually is provided between the thickeners and the flotation roughers. At some plants alkalinity is determined by titration with standard acid; at others the degree of alkalinity expressed as the pH value is determined. At Sherriitt-Gordon and Noranda, soda ash rather than lime is used in conjunction with cyanide and zinc sulphate for the depression of pyrite, pyrrhotite and sphalerite.

Collectors. - The usual collectors employed at present in copper concentrators are xanthate and aerofloat, the former being used more commonly than the latter. A small amount of collector and a high alkalinity tend to depress pyrite and yield a relatively high concentration ratio, while a large amount of collector and a low alkalinity tend to float pyrite, yielding a relatively low ratio of concentration. As a rule, the collector is added to the ore pulps of selective copper-flotation circuits between the grinding and flotation circuits. The amounts of xanthate or aerofloat used at 21 concentrators ranged from 0.017 to 0.409 pound and averaged 0.076 pound per ton of ore treated.

Frothers. - Pine oil and cresylic acid are the usual frothing reagents employed at copper concentrators, the former being used more generally than the latter. The frothing reagent is usually added to the ore pulps between the grinding circuit and the flotation roughers, additional amounts being added as required in the flotation circuit. The amounts of frothing reagent used at 21 concentrators ranged from 0.040 to 0.350 pound and averaged 0.180 pound per ton of ore treated.

Miscellaneous

Primary Slime

For some time it has been recognized that the primary mine slime contained in feeds to some copper concentrators is more difficult to treat by flotation than freshly ground ore. If the surfaces of copper sulphide minerals contained in primary mine slime are tarnished as compared to freshly ground sulphide minerals due either to this condition existing in the unbroken ore or to exposure in stopes or bins after breaking in the mine, then it appears probable that flotation conditions could be varied to advantage in the separate conditioning or separate treatment of primary slime. Moreover, the segregation of oxidized copper minerals in primary slime is well-known, and it might be advantageous to treat the slime separately for this reason. It is also generally accepted that although the tarnished sulphides and oxidized copper minerals tend to segregate in the primary slime and the first slime produced in crushing and grinding, on the other hand, pyrite tends to segregate in the classifier sands of secondary grinding circuits especially when an open primary grinding

circuit is employed. The general conditions for the treatment of primary and secondary classifier overflows therefore differ in that the minerals of the primary classifier overflow are slower in floating than those of the secondary overflow containing freshly ground sulphides; furthermore, the presence of freshly ground copper sulphides and a higher pyrite content in secondary classifier overflows suggest a rapid floating of the copper minerals if pyrite is to be depressed. At some concentrators flotation results have been improved recently by either separate conditioning or separate conditioning and separate treatment of primary mine slime more or less diluted by the slimes produced in crushing and primary grinding operations. At other plants the results of large-scale tests indicate that special treatment of primary slime does not materially improve results.

Tables

Tables in copper concentrators treating sulphide ores have been eliminated generally owing to the finer grinding employed, which permits a satisfactory recovery with flotation alone and to the general adoption of selective-flotation methods with consequent depression of pyrite.

Gravity concentration with jigs and tables is, of course, still used for most of the recovery in the Lake Superior district owing to the toughness of native copper in the ores treated, as previously noted. In this district flotation is an adjunct to gravity-concentration methods.

At the concentrator of the Magma Copper Co., tables are used to rough out part of the sulphides after primary grinding to 12-mesh. The Magma concentrator feed has a higher copper content than average concentrating ore; it averaged 7.12 percent of copper for the first 6 months of 1929. Tables recovered 39.53 percent of the total copper during this period, reducing the copper content in the flotation feed to 5.23 percent. Moreover, at Magma it is desired to recover as much pyrite as possible for the fluxing of high-grade siliceous mine ores at the smelter and incidentally for the gold and silver values associated with the pyrite.

At the concentrator of the International Nickel Co. tables are used intermittently as scavengers following flotation, as it is desired to recover all sulphides in order to reduce nickel and precious-metal losses in the tailing. MacDonald⁵⁰ states that the concentrator is being operated alternately with and without tables to determine whether the use of tables is justified.

Microscopic Analyses

The use of the microscope for the examination of concentrator products is increasing. In a total of 23 concentrators considered, the microscope is used for routine work at 7 and samples of concentrator products are occasionally sent out for microscopic analysis at 6. The method commonly used to prepare

⁵⁰/ MacDonald, W. T., Selective Flotation Mill at Copper Cliff: Eng. and Min. Jour., vol. 130, Nov. 10, 1930, p. 470.

samples of concentrator products for microscopic examination consists of sizing the product to be examined and embedding each size in a matrix, usually bakelite, under pressure. A surface of the resulting briquet is ground on a revolving iron wheel and then on a stationary glass plate with fine abrasives. The ground surface is finally polished with fine abrasive on a rapidly revolving wheel covered with muslin, linen, or broadcloth.

Microscopic examination of the polished surface identifies the copper minerals contained in the material and indicates the association of each mineral to the gangue and other sulphide minerals. Routine work is usually qualitative, but monthly composite samples of concentrator products are examined quantitatively at some plants, which enables changing characteristics in concentrator feed to be recognized.

Although the microscopic examination of concentrator products yields information not obtainable by chemical methods, certain errors are inherent in the method and caution is essential in the interpretation of results. For example, a grain of chalcocite attached to gangue may be embedded in bakelite in such a position that the polished surface does not reveal the gangue, and the chalcocite under those conditions may be reported as free chalcocite.

Notwithstanding admitted errors in the method when used quantitatively, the microscope has proved an additional and valuable tool in designing new flow sheets and remodeling old ones. In recent remodeling operations at the Cananea concentrator, which resulted in greatly improved metallurgical results and reduced costs, A. T. Tye in a private communication to the author attributes the success attained largely to microscopic analyses made by Gaudin.

Overgrinding

The effect of degree of grinding of copper ores on recovery and grade of concentrate has been discussed in many of the Information Circulars on milling methods recently published by the Bureau of Mines. With present degrees of grinding used most of the copper lost in tailings is usually contained in the minus 200-mesh sizes due not only to the preponderance of material of these sizes in the tailing but also to the loss of appreciable amounts of fine free-copper minerals in these sizes. The fact that oxidized copper minerals are known to segregate in the finer screen sizes does not account for the total loss of fine free material in slimes, as much of the loss is found by microscopic analyses to be finely divided free-sulphide minerals. Some attention has been given lately to the effect of overgrinding copper ores, especially upon recovery, and while the extent of loss suffered by overgrinding is not generally known the fact that grinding beyond a certain fineness does increase losses is known to apply to some ores.

CONCENTRATING COSTS

Table 4 gives a summary of costs classified by subdivisions of the concentrating process and table 5 a summary of costs segregated into labor, power, and supplies. In these tables the plants have been arranged in order of tonnage handled per day for the period covered.

Distributions of power used in concentrating at typical plants are given in table 6 and the quantities of water used, summaries of ore treated per man-shift, and unit power costs in table 7. Distributions of labor in terms of ore treated per man-shift, classified into subdivisions of the concentrating process, are presented in table 8 for typical plants.

TABLE 4. - Operating costs at typical copper concentrators ^a/₁

Concentrator	Period covered		Ore milled per day, tons	Operating costs per ton of ore milled, dollars				
	Year	Number of months		Coarse crushing	Fine crushing	Total crushing	Grinding and classifying	Flotation
Magna and Arthur, Utah Copper Co.	1929	1	42,493	\$0.0197	\$0.0450	\$0.0647	\$0.0918	\$0.0914
Miami Copper Co. Hayden, Nevada	1930	12	17,141			.0581	.0913	.0766
Cons. Copper Co.	1929	12	8,862	.0298	.0812	.1110	.0778	.0940
Britannia Mining & Smelting Co., Ltd.	1931	1	6,436	.0052	.0486	.0538	.0460	.0377
Morenci, Phelps Dodge Corp.	1930	1	5,014	.039	.068	.107	.125	³ / ₁ .084
Copper Queen, Phelps Dodge Corp.	1929	12	4,251	.059		.059	.194	.075
Calumet & Hecla Cons. Copper Co.	1929	12	2,900		⁴ / ₁ .192		.081	⁵ / ₇ .072 ⁷ / ₈ .039 ⁸ / ₉ .101
Nacozari, Phelps Dodge Corp.	1928 and 1929	24	⁹ / ₁ 2,516	.0615		.0615	.2229	.1053
Cananea Cons. Copper Co.	1929	6	2,400	.0373		.0373	.1912	.1310
United Verde Copper Co.	1929	1	1,532	.0751	.1180	.1931	.1639	.1757
Old Dominion, Phelps Dodge Corp.	1929	12	1,338	.086	.064	.150	.187	.140
Engels Copper Mining Co.	1929	12	1,091	.0856	.0275	.1131	.2852	.0932
Minas de Matambre, S. A. ..	1930	12	¹¹ / ₁ 1,000	¹² / ₁ .037	.093	¹² / ₁ .130	.279	.154
Sheritt Gordon Mines, Ltd. ¹ / ₁ ..	1931	6	838	.0173	.0568	.0741	.3122	¹⁴ / ₁₅ .2415
Magma Copper Co.	1929	6	630			.0816	.2972	¹⁴ / ₁₅ .1749
Verde Central Mines, Inc.	1929	1	392			.377	.338	¹⁵ / ₁ .0265 .154

^a See page 36 for footnotes.

TABLE 4. - Operating costs at typical copper concentrators^{1/} --Continued

Operating costs per ton of ore milled, dollars							
Concentrator	Dewater- ing con- centrate ^{2/}	Sampling and assaying	Experi- mental	Disposal of tailing	Water	Miscel- laneous	Total
Magna and Arthur, Utah Copper Co.	\$0.0052	\$0.0042	\$0.0072	\$0.0142	\$0.0100	\$0.0135	\$0.3022
Miami Copper Co. Hayden, Nevada	.0059			.0123	.0237	.0288	.2967
Cons. Copper Co.	.0173	.0097	.0094	.0140	.0232	.0426	.3990
Britannia Mining & Smelting Co., Ltd.	.0094	.0020				.0281	.1770
Morenci ~~Ph~~ elps Dodge Corp.	.022	.014	.003	.011	.082	.052	.500
Copper Queen, Phelps Dodge Corp.	.019	.014		.022	.059	.051	.493
Calumet & Hecla Cons. Copper Co.				.017		.031	^{6/} .533
Nacozari, Phelps Dodge Corp.	.0170	.0137		.0276	.0179	.0471	.5130
Cananea Cons. Copper Co.	.0154	.0069		.0076	.0392	.1887	.6173
United Verde Copper Co.	.0336		^{12/} .0331	.0046	.0275	.1050	.7365
Old Dominion, Phelps Dodge Corp.	.042	.044		.025	.029	.082	.699
Engels Copper Mining Co.	.0329	.0117		.0583	.0014	.0563	.6521
Minas de Mata- hambre, S. A. ..	.033			.019		.132	.747
Sherritt Gordon Mines, Ltd. ^{1/}	.0564	.0187		.0144			^{13/} .7173
Magma Copper Co. Verde Central Mines, Inc.	.0052 .054	.0497 .018		.0135 .082	.0222 .032	.2634 .137	.9342 1.192

^{1/} All data taken from information circulars, U. S. Bureau of Mines except that for Sherritt Gordon Mines, Ltd. for which see Mott, R. C., The Sherritt Gordon Mill; Canadian Min. Jour.; March 1932, p. 109. ^{2/} Includes handling after dewatering in most cases. ^{3/} Includes \$0.008 per ton for operation of tables. ^{4/} Steam stamps. ^{5/} Gravity concentration, entire tonnage. ^{6/} Coarse crushing not included. ^{7/} Flotation of slimes. ^{8/} Leaching of sands. ^{9/} For last six months of 1929. ^{10/} Routine sampling and testing. ^{11/} Approximate. ^{12/} Includes \$0.010 per ton for sorting. ^{13/} An additional item of \$0.1065 per ton is charged against milling as "Replacement reserve." ^{14/} Flotation. ^{15/} Tabling.

TABLE 5. - Operating costs at typical copper concentrators ^a 1/

Concentrator	Period covered		Ore milled per day, tons	Operating costs per ton of ore milled, dollars			
	Year	Number of months		Labor			Power
				Operating	Repairs	Total	
Magna and Arthur, Utah Copper Co.	1929	1	48,493	\$0.0429	\$0.0233	\$0.0662	\$0.1104
Miami Copper Co.	1930	12	17,141	.0495	.0325	.0820	.0654
Hayden, Nevada Cons. Copper Co.	1928	12	8,862	.0689	.0363	.1052	.1037
Britannia Mining & Smelting Co., Ltd.	1931	1	6,486	.0500	.0185	.0685	.0250
Morenci, Phelps Dodge Corp.	1930	1	5,014			.174	.115
Copper Queen, Phelps Dodge Corp.	1929	12	4,281			.151	.121
Nacozari, Phelps Dodge Corp.	1928 and 1929	24	<u>2</u> / 2,516	<u>3</u> / .0877	<u>3</u> / .0494	.1371	.1385
Cananea Cons. Copper Co.	1929	6	2,400	.0872	.0322	.1194	.2034
United Verde Copper Co. <u>4</u> /	1929	1	1,532	.1564	.0239	.1803	.1233
Old Dominion, Phelps Dodge Corp.	1929	12	1,338	.157	.063	.220	.173
Engels Copper Mining Co.	1929	12	1,091	.1715	.0426	.2141	.1578
Minas de Matahambre, S. A. ..	1930	12	<u>5</u> / 1,000				
Sherritt Gordon Mines, Ltd.	1931	6	838	.1052	.0188	.1240	.1702
Magma Copper Co.	1929	6	650	.1527	.1045	.2572	.1935
Verde Central Mines, Inc.	1929	1	392			.524	.291

^a See page 38 for footnotes.

TABLE 5. - Operating costs at typical copper concentrators^{1/} -Continued

Operating costs per ton of ore milled, dollars					
Concentrator	Supplies			Miscellaneous	Total
	Other than reagents	Reagents	Total		
Magna and Arthur, Utah Copper Co.	\$0.0724	\$0.0353	\$0.1077	\$0.0178	\$0.3021
Miami Copper Co.	.0782	.0397	.1179	.0315	.2968
Hayden, Nevada Cons. Copper Co.			.1350	.0551	.3990
Britannia Mining & Smelting Co., Ltd.	.0668	.0167	.0835		.1770
Morenci, Phelps Dodge Corp.			.133	.078	.500
Copper Queen, Phelps Dodge Corp.	.119	.042	.161	.060	.493
Nacozari, Phelps Dodge Corp.	.1575	.0764	.2339	.0035	.5130
Cananea Cons. Copper Co.	.2004	.0554	.2558	.0387	.6173
United Verde Copper Co. ^{4/}	.0806	.1249	.2055	.0248	^{4/} .5339
Old Dominion, Phelps Dodge Corp.	.152	.069	.221	.085	.699
Engels Copper Mining Co.	.2324	.0478	.2802		.6521
Minas de Matahambre, S. A.					.747
Sherritt Gordon Mines, Ltd.			.3618	.0613	.7173
Magna Copper Co.	.2325	.0622	.2947	.1888	.9342
Verde Central Mines, Inc.			.377		1.192

^{1/} All data taken from information circulars, U. S. Bureau of Mines, except that for Sherritt Gordon Mines, Ltd., for which see Mott, R. C., The Sherritt Gordon Mill; Canadian Min. Jour., Mar. 1932, p. 109. ^{2/} For last 6 months of 1929. ^{3/} Distribution of total labor cost into operation and repairs computed by author. ^{4/} Coarse and fine crushing not included. ^{5/} Approximate.

TABLE 6. - Distribution of power at typical copper concentrators ^a/₁

Concentrator	Period covered		Ore mill- ed per day, tons	Distribution of power, kilowatt-hours per ton of ore milled			
	Year	Number of months		Crushing			Grinding and Classifying
				Coarse	Fine	Total	
Magna and Arthur, Utah Copper Co.	1929	1	48,493	0.61	2.89	3.50	7.51
Miami Copper Co. Hayden, Nevada	1930	12	17,141			1.87	6.13
Cons. Copper Co.	1928	12	8,862	.44	3.24	3.68	3.74
Britannia Mining & Smelting Co., Ltd.	1931	1	6,486	.10	2.60	2.70	7.81
Morenci, Phelps Dodge Corp.	1930	1	5,014			2.80	7.54
Copper Queen, Phelps Dodge Corp.	1929	12	4,281	1.05		1.05	9.01
Nacozari, Phelps Dodge Corp.	1928 and 1929	24	2,516	.82		.82	11.42
Cananea Cons. Copper Co.	1929	6	2,400	1.05		1.05	7.66
United Verde Copper Co.	1929	1	1,532				9.80
Old Dominion, Phelps Dodge Corp.	1929	12	1,338	1.38	1.07	2.45	8.89
Engels Copper Mining Co.	1929	12	1,091	2.26	1.19	3.45	12.15
Minas de Mata- hambro, S. A. ..	1930	12	<u>4</u> / 1,000			2.50	
Magma Copper Co.	1929	6	650			3.36	6.38

^a See page 40 for footnotes.

TABLE 6. - Distribution of power at typical copper concentrators^{1/} -Continued

Concentrator	Distribution of power, kilowatt-hours per ton of ore milled					
	Flotation	Dewatering concentrate	Disposal of tailing	Initial water	Miscellaneous	Total
Magna and Arthur, Utah Copper Co.	4.92	0.18		0.34	0.65	17.10
Miami Copper Co.	2.35		0.25		.94	11.54
Hayden, Nevada Cons. Copper Co.	1.76	.24		1.28	.35	11.05
Britannia Mining & Smelting Co., Ltd.	1.44	^{2/} .71			.18	12.84
Morenci, Phelps Dodge Corp.	1.46				2.36	14.16
Copper Queen, Phelps Dodge Corp.	1.66	.43	..	1.22	.33	13.70
Nacozari, Phelps Dodge Corp.	1.13	.48	.82	1.00	.57	16.24
Cananea Cons. Copper Co.	3.72	.39		1.99	.46	15.27
United Verde Copper Co.	4.23	.90		.16	1.26	^{3/} 16.35
Old Dominion, Phelps Dodge Corp.	2.67	.78	.74		.43	15.96
Engels Copper Mining Co.	3.65	.44	.17		.09	19.95
Minas de Mata-hambré, S. A. ..					^{5/} 15.73	18.23
Magma Copper Co.	6.65	.08		1.46	^{6/} 1.36	19.29

^{1/} All data taken from information circulars, U. S. Bureau of Mines. ^{2/} Dewatering and handling concentrate. ^{3/} Coarse and fine crushing not included. ^{4/} Approximate. ^{5/} Total milling except primary crushing. ^{6/} Includes 0.42 kw.-hr. for table operation.

TABLE 7. - Water, labor, and unit power costs at
typical copper concentrators a 1/

Concentrator	Period covered		Ore mill- ed per day, tons	Water per ton of ore, gallons		
	Year	Number of months		New	Reclaimed	Total
Magna and Arthur, Utah Copper Co.	1929	1	48,493	884		884
Miami Copper Co.	1930	12	17,141	137	355	492
Hayden, Nevada Cons. Copper Co.	1928	12	8,862	726		726
Britannia Mining & Smelting Co., Ltd.	1931	1	6,486	1,300		1,300
Morenci, Phelps Dodge Corp.	1930	1	5,014	283	1,275	1,558
Copper Queen, Phelps Dodge Corp.	1929	12	4,261	275	1,175	1,450
Calumet and Hecla Cons. Copper Co.	1929	12	2,900			
Nacozari, Phelps Dodge Corp.	1929	6	2,516	524	669	1,193
Cananea Cons. Copper Co.	1929	6	2,400			
United Verde Copper Co.	1929	1	1,532	367	336	703
Old Dominion, Phelps Dodge Corp.	1929	12	1,338	1,105		1,105
Engels Copper Mining Co.	1929	12	1,091	680		
Minas de Mata- hambre, S. A. ..	1930	12	2/1,000	167	500	667
Magma Copper Co.	1929	6	650	300	1,100	1,400
Verde Central Mines, Inc.	1929	1	392	417	500	917

a See page 42 for footnotes.

TABLE 7. - Water, labor, and unit power costs at
typical copper concentrators^{1/}--Continued

Concentrator	Labor ore milled per man-shift, tons			Unit cost of power per kilowatt hour, cents
	Operation	Maintenance	Total	
Magna and Arthur, Utah Copper Co.				0.663
Miami Copper Co.	102.5	250.6	72.7	.567
Hayden, Nevada Cons. Copper Co.			28.2	.938
Britannia Mining & Smelting Co., Ltd.	108	295	80.0	.200
Morenci, Phelps Dodge Corp.	50.4	75.0	30.1	.812
Copper Queen, Phelps Dodge Corp.	37.4	189.6	31.2	.883
Calumet & Hecla Cons. Copper Co.				.580
Nacozari, Phelps Dodge Corp.	<u>2/</u> 30.1	<u>2/</u> 54.3	<u>2/</u> 19.4	<u>2/</u> .804
Cananea Cons. Copper Co.	22.5	76.7	15.9	1.330
United Verde Copper Co.	46.3	239.8	38.8	.750
Old Dominion, Phelps Dodge Corp.	36.7	92.3	26.4	1.004
Engels Copper Mining Co.	27.4	114.9	22.2	.791
Minas de Mata- hambre, S. A.				
Magma Copper Co.	34.2	81.3	24.1	.964
Verde Central Mines, Inc.				1.500

^{1/} All data taken from information circulars, U. S. Bureau of Mines.^{2/} Average of 1928 and 1929.^{3/} Approximate.

TABLE 8. - Distribution of labor at typical copper concentrators ^a

	Miami Copper Co. ^{1/}			Britannia Mining & Smelting Co., Ltd. ^{2/}		
	Mill treated 17,140 tons per day, 1930			Mill treated 6,486 tons per day, July, 1931		
	Tons per man-shift			Tons per man-shift		
	Operation	Mainte- nance	Total	Operation	Mainte- nance	Total
Coarse and fine crushing..	403	1,033	297	202		
Grinding and classifying..	494	972	323	1,081		
Flotation.....	773	1,830	543	1,622		
Dewatering and handling concentrate.....	1,986	8,403	1,607	1,622		
Sampling and assaying.....	3/ 952	3/ 1,120	3/ 878	3,243		
Disposal of tailing.....	227	4,529	1,438			
Water supply	2,751	1,926	1,133			
Experimental	5,340		5,340	6,486		
Miscellaneous.....	721	1,894	522	463		
Total.....	102	251	73	108	295	80

	Nacozari, Phelps Dodge Corp. ^{4/}			United Verde Copper Co. ^{5/}		
	Mill treated 2,516 tons per day 1929			Mill treated 1,532 tons per day, September, 1929		
	Tons per man-shift			Tons per man-shift		
	Operation	Mainte- nance	Total	Operation	Mainte- nance	Total
Coarse and fine crushing..	164			6/		
Grinding and classifying..	166			344	668	227
Flotation.....	393			286	1,394	237
Dewatering and handling concentrate	463			269	1,333	224
Sampling and assaying.....	421			2/ 1,229	2/ 1,743	7/ 227
Disposal of tailing.....	286			1,532	8,481	1,298
Water supply.....	565			4,444	6,054	2,563
Experimental.....	1,084					
Miscellaneous.....	127			160	1,124	140
Total.....	30	54	19	8/46	8/240	8/39

^a See page 45 for footnotes.

TABLE 8. - Distribution of labor at typical copper concentrators--Continued^a

	Old Dominion Phelps Dodge Corp. ^{9/}			Engels Copper Mining Co. ^{10/}		
	Mill treated 1,338 tons per day, 1929			Mill treated 1,091 tons per day, 1929		
	Tons per man-shift			Tons per man-shift		
	Operation	Mainte- nance	Total	Operation	Mainte- nance	Total
Coarse and fine crushing			94	110	377	85
Grinding and classifying			157	11/133	11/535	11/107
Flotation.....			253	234	1,333	199
Dewatering and handling concentrate			291	266	5,000	252
Sampling and assaying...			956	1,639	20,000	1,515
Disposal of tailing.....			382	147	1,075	129
Water supply.....			6,690	3,448		3,448
Experimental.....						
Miscellaneous.....			104	243	444	157
Total.....	37	92	26	27	115	22

	Magma Copper Co. ^{12/}		
	Mill treated 650 tons per day, last half of 1929		
	Tons per man-shift		
	Operation	Maintenance	Total
Coarse and fine crushing.....	228	765	176
Grinding and classifying.....	327	178	115
Flotation.....	183	699	144
Dewatering and handling concentrate..	...	4,333	4,333
Sampling and assaying.....	174	1,857	159
Disposal of tailing.....	524	3,250	451
Water supply.....	...	1,757	1,757
Experimental.....	...		
Miscellaneous.....	116	436	92
Total.....	34	81	13/24

^a See page 45 for footnotes.

- 1/ After Hunt, H. D., Milling Methods and Costs at the Concentrator of the Miami Copper Co., Miami, Ariz.: Inf. Circ. 6573, Bureau of Mines, 1932, p. 34.
- 2/ Munro, A. C., and Pearce, H. A., Milling Methods and Costs at the Concentrator of the Britannia Mining and Smelting Co., Ltd., Britannia Beach, B. C.: Inf. Circ. 6619, Bureau of Mines, 1932, p. 24.
- 3/ Sampling only.
- 4/ After Rose, E. H., and Cramer, W. B., Milling Methods and Costs at the Nacozari Concentrator of the Phelps Dodge Corporation, Nacozari, Sonora, Mexico: Inf. Circ. 6353, Bureau of Mines, 1930, p. 37.
- 5/ After Kuzell, C. R., and Barker, L. M., Milling Methods and Costs at the Concentrator of the United Verde Copper Co., Clarkdale, Ariz.: Inf. Circ. 6343, 1930, p. 28.
- 6/ Crushing plant at smelter.
- 7/ Sampling and testing.
- 8/ Crushing not included.
- 9/ After Forrester, D. L., and Cramer, W. B., Milling Methods and Costs at the Concentrator of the Old Dominion Co., Globe, Ariz.: Inf. Circ. 6467, Bureau of Mines, 1931, p. 29.
- 10/ After Nelson, W. I., Milling Methods and Costs at the Superior Concentrator of the Engels Copper Mining Co., Plumas County, Calif.: Inf. Circ. 6550, Bureau of Mines, 1932, p. 22.
- 11/ Classifying not included.
- 12/ After Rose, J. H., and McNabb, J. C., Milling Methods and Costs at the Concentrator of the Magma Copper Co., Superior, Ariz.: Inf. Circ. 6319, Bureau of Mines, 1930, p. 19.
- 13/ Superintendence and watchmen not included.